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Dewatering Active Underground Coal Mines. Technical Aspects and Cost-Effectiveness

W. A. Wahler and Associates, Palo Alto, CA

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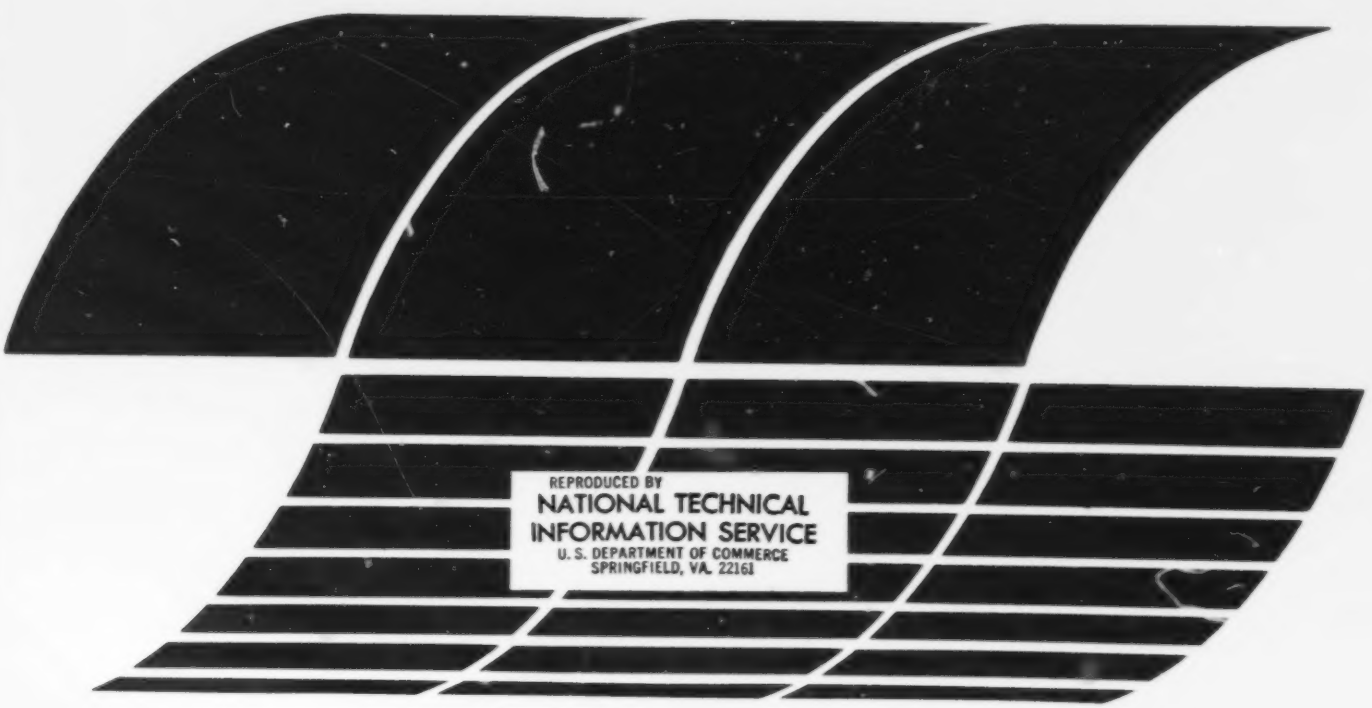
Technical Aspects and Cost-Effectiveness

Interagency Energy/Environment R&D Program Report



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DEWATERING ACTIVE UNDERGROUND COAL MINES: TECHNICAL ASPECTS AND COST-EFFECTIVENESS

by

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Contract No. 68-03-2366

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FOREWORD

When energy and material resources are extracted, processed, converted, and used, the related pollutional impacts on our environment and even on our health often require that new and increasingly more efficient pollution control methods be used. The Industrial Environmental Research Laboratory - Cincinnati (IERL-Ci) assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

One possible means of controlling acid mine drainage is to intercept groundwater before it actually enters the active coal mine. This report presents the technical results of a pilot-scale dewatering operation and assesses the cost-effectiveness of the technique. The study should provide the coal mining industry with useful concepts and general guidance in determining the technical and economical feasibility of thus dewatering an active coal mine. The study also suggests measures that may be cost-effective in controlling mine water quality with typical mine water removal systems. The practical geohydrologic experience with wells gained during this study should be useful to others involved with mine dewatering research and also points out areas where additional work is needed.

Further information on this subject may be obtained from the Extraction Technology Branch.

David G. Stephan
Director
Industrial Environmental Research Laboratory
Cincinnati

ABSTRACT

This study evaluated the cost-effectiveness of dewatering an active underground coal mine as an alternative or supplement to treating acid mine drainage. Through a combination of research, personal contacts, and site visits to selected wet mines in the northern part of the eastern bituminous coal mining region, the Barnes and Tucker Company's Lancashire No. 20 mine, located in western Pennsylvania, was selected as a suitable site for a pilot-scale dewatering operation. A dewatering program was formulated and base-line data collection was performed in conjunction with exploration of hydrogeologic conditions and well construction. Dewatering was then performed for a total of 25 days to provide the basis for the technical and cost analysis.

The results of well testing and pilot dewatering indicated that groundwater flow into the mine is controlled by fracture zones, a condition that appeared to be prevalent in the wet mines inspected during this study. The hydraulic characteristics of this type of flow system are difficult to define, and a study is currently being performed by D'Appolonia Consulting Engineers, Inc. to locate and define the quantities of water that enter underground mines. However, the dewatering operation demonstrated that mine inflows could be intercepted (well-effectiveness was projected to range from 50 to 80 percent) and could reduce the amount of water degraded in the mine. Though individually pumped wells constructed from the surface proved not to be cost-effective at the study mine, this system could be cost-effective (without considering other benefits such as roof stability), if the acidity of the mine drainage were higher (ranging from 500 to 1500 mg/ℓ as $CaCO_3$), and if the intercepted groundwater could be discharged without treatment. A more economical system (which would be cost-effective at the study mine) appears to be the use of angled drain holes drilled from the mine openings; costs for this system are not as sensitive to drainage quality or mine depth. Monitoring of water quality as it was transferred through the mine indicated that degradation is a gradual process; therefore closed discharge systems or more direct transfer to the surface might also prevent substantial water degradation.

This report was submitted in fulfillment of Contract No. 68-03-2366 by W. A. Wahler & Associates, of Palo Alto, California, under the sponsorship of the U.S. Environmental Protection Agency. This report covers the period December 8, 1975 to June 30, 1978, and work was completed as of August 30, 1978.

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SYMBOLS AND ABBREVIATIONS

SI Metric		Customary U.S. Measure	
Symbol	Unit	Abbrev.	Unit
m	meter	in.	inch(es)
nm	nanometer	ft	foot (feet)
cm	centimeter	yd	yard(s)
km	kilometer	cu yd	cubic yard(s)
m^2	square meter	mi	mile(s)
ha	hectare	gal	gallon(s)
m^3	cubic meter	lb	pound(s)
s	second	ton	short ton(s)
d	day	sec	second(s)
m/s	meters per second	h	hour(s)
m^3/s	cubic meters per second	day	day(s)
$m^3/m \cdot d$	cubic meters per meter per day	gpm	gallons per minute
ℓ	liter	mgd	million gallons per day
ℓ/s	liters per second	hp	horsepower
g	gram		
mg	milligram		
kg	kilogram		
mg/ℓ	milligrams per liter		
t	tonne		
W	watt		
kW	kilowatt		
µS/cm	microsiemens per centimeter		

CONVERSION TABLE

Data are reported in this document primarily in SI metric units. Customary U.S. units are also given in parentheses to make the data fully accessible to sections of industry which have not yet undertaken metrication. Where the data are reported from (or compared with) earlier studies which did not use SI metric units, the customary U.S. units are used as the primary expression of measurement.

SI Metric		U.S. Equivalent	
1 kilometer (1000 m)	=	0.621	mile
	=	3280.84	feet
	=	1093.61	yards
1 meter	=	39.370	inches
	=	3.281	feet
	=	1.094	yards
1 centimeter (1×10^{-2} m)	=	0.394	inch
	=	3.281×10^{-2}	foot
1 nanometer (1×10^{-9} m)	=	39.370×10^{-9}	inch
	=	3.281×10^{-9}	foot
1 square meter	=	1550.16	square inches
	=	10.765	square feet
1 hectare (10 000 m^2)	=	3.861×10^{-3}	square mile
1 cubic meter	=	6.102×10^{4}	cubic inches
	=	2.642×10^{2}	gallons (U.S. fluid)
	=	35.314	cubic feet
	=	1.308	cubic yards
1 liter (1×10^{-3} m^3)	=	0.264	gallon (U.S. fluid)
	=	35.314×10^{-3}	cubic foot
1 gram	=	2.205×10^{-3}	pound (avoirdupois)
1 milligram	=	2.205×10^{-6}	pound (avoirdupois)
1 kilogram	=	2.205	pounds (avoirdupois)

CONVERSION TABLE (continued)

SI Metric		U.S. Equivalent	
1 tonne (1000 kg)	=	2.205×10^3	pounds (avoirdupois)
	=	1.102	short tons
1 meter per second	=	3.281	feet per second
	=	2.237	miles per hour
1 cubic meter per second	=	35.314	cubic feet per second
	=	2.282×10^7	gallons per day
	=	22.82	million gallons per day
	=	1.585×10^4	gallons per minute
1 cubic meter per day	=	35.314	cubic feet per day
	=	2.642×10^2	gallons per day
1 cubic meter per meter per day	=	10.765	cubic feet per foot per day
1 liter per second	=	35.314×10^{-3}	cubic foot per second
	=	15.850	gallons per minute
	=	2.282×10^4	gallons per day
1 watt	=	1.340×10^{-3}	horsepower
1 kilowatt (1000 watts)	=	1.340	horsepower
1 microsiemen per centimeter	=	1	micromho per centimeter

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Under subcontract to W. A. Wahler & Associates, Environmental Control Technology Corporation (ENCOTEC) of Ann Arbor, Michigan provided services in water quality monitoring and chemical analyses, and Chase-Mogdis, Inc. of Ann Arbor, Michigan provided services in economic analyses and cost comparisons. Toy Drilling Company of Distant, Pennsylvania subcontracted drilling and pumping services.

The personnel of the Lancashire No. 20 mine near Carrolltown, Pennsylvania, owned and operated by the Barnes and Tucker Company, provided valuable assistance to the study team. Despite the inconveniences imposed upon them by the study, they were most cooperative in supplying mine data, cost data, and water quality information, in surveying drill holes, and in providing general assistance underground, including recording flow measurements and collecting water samples. We are particularly indebted to Mr. George Merritts, Chief Engineer and Mr. Jerry Sousa, Mining Engineer, for their active support, patience, and sustained interest. We owe thanks also to Mr. Robert W. Roland, Assistant Vice President of Finance of the Barnes and Tucker Company, for his cooperation and for his provision of costing data.

The U. S. Bureau of Mines and various state agencies contributed information and advice in a friendly and cooperative manner. The Department of Environmental Resources and the Bureau of Water Quality of the Commonwealth of Pennsylvania were of particular help. Valuable guidance was provided by the staff of the U.S. Environmental Protection Agency, Industrial Environmental Research Laboratory in Cincinnati, Ohio, in the administration and performance of the study. Special recognition is due to Mr. S. Jackson Hubbard, Project Officer.

SECTION 1

INTRODUCTION

Acid mine drainage (AMD) is a major pollution problem in the eastern coal mining regions. In Pennsylvania alone, Klingensmith (1965)[1] estimates 3700 km (2,300 mi) of streams have been adversely affected. Underground mines contribute a large portion of the acid drainage water which constitutes a widespread pollution problem throughout the eastern coal mining regions. In underground mines AMD is formed when iron disulfides (e.g. marcasite, pyrite) decompose on exposure to air, forming a complex series of compounds that are highly soluble in water. Groundwater inflow to the mine, which may be of excellent or poor quality when it enters the mine, dissolves these compounds, forming acid water characteristically high in iron.

Natural groundwater flow patterns are changed by an underground coal mine, which acts as a large subsurface drain or sink, inducing flow into the mine. The inflow rates are dependent on the hydraulic characteristics of the surrounding rock units, the degree of fracturing of these units, and the hydraulic gradients induced by drainage into the mine. Several types of rock can act as primary aquifers that store and transmit groundwater. In the bituminous coal region of the eastern United States, interbedded sandstones, shales, siltstones, and some limestones associated with the coal strata serve these functions. In particular, sandstone channels may be sources of large inflows where they are in contact with the mine opening and have a relatively high permeability. Where fracture-dominated flows occurs, all of these rock types can act as secondary aquifers and provide storage space to recharge fracture zones.

The formation of AMD has been extensively studied in recent years along with various techniques for water quality control. Water quality control can be achieved either by treating drainage water outside the mine or by preventing contact of air or water with the iron disulfides. Treatment, usually by neutralization, is the most common method currently in use, but it is expensive and not too satisfactory for various reasons. In particular, the addition of calcium in solution tends to increase total dissolved solids and hardness of the water and the sludge generated may be difficult to dispose of in a safe manner. Prevention of air contact with acid-forming minerals--for example, by using remote controlled mining methods--holds some promise for the future. Water contact should be controllable by interception of groundwater inflows to the mine (dewatering).

Interception of groundwater inflows to active coal mines is the subject of this study, which has involved the construction and operation of a pilot well dewatering system to collect and analyze both technical and cost data. The Lancashire No. 20 mine, located near Carrolltown, Pennsylvania, was the

study site for the pilot dewatering program. This site was selected for several reasons: the mine inflows are not only easily monitored, but are of high quality in contrast to the mine drainage, which requires treatment before it can be discharged; and the owner, the Barnes and Tucker Company, was very cooperative and interested in the study.

The objectives of the pilot study were (1) to determine the impact of the dewatering operation on the quantity and quality of inflows in the limited area of the mine used for study; (2) to evaluate, by extrapolating the results of this program, the potential effectiveness of the dewatering technique for the mine as a whole; and (3) to perform an economic evaluation of the technique. Consideration of several potential sites for the study and review of information available in current literature, State and Federal agencies, and industry provided material for technical and economic comparisons.

SECTION 2

CONCLUSIONS

The following is a summary of the major conclusions reached from the results of this study.

1. Though several of the rock strata that occur in association with coal measures can act as aquifers, the inspection of a limited number of mines during this study suggests that fracture-dominated flow systems are prevalent where large mine inflows of groundwater are experienced. The larger inflows are usually localized along fracture zones associated with faulting and/or jointing, which in turn are often associated with folds.

2. Exploration and pump tests of wells revealed that groundwater flows into the study area of the mine were concentrated along two fracture zones that appear to intersect in the area of greatest inflow. These preferred and rather narrow flow paths were not revealed until after completion of observation wells and dewatering wells. This late recognition indicated a need for better exploration techniques to recognize and define the characteristics of such flow paths, as discussed in Section 3.

3. Baseline monitoring of water quantities established that flows respond rapidly to wet and dry periods in spite of the depth of the study mine--over 152 m (500 ft)--and thus indicated that fracture zones have hydraulic connection to the surface. Baseline water quality data indicate that degradation of mine waters occurs gradually as the water is transferred through the mine; thus the water does not require treatment until it reaches the last few transfer stations.

4. The quality of water pumped from dewatering wells remained high, permitting direct discharge to Laurel Lick Run. In some respects the water quality was higher than the water quality generally occurring in the receiving streams. The intercepted groundwater would be of benefit to stream water quality because of the higher alkalinity and pH which would add buffering capacity to the creek. Therefore, large discharges from a full-scale dewatering system could benefit a stream in this manner, making it less sensitive to degradation by acid drainage.

5. Pilot-scale dewatering, after several trials and interruptions, was performed continuously for 14 days, with three wells pumping an average of 5.2 ℓ/s (82 gpm). The inflow to the study area was reduced from 6.94 ℓ/s (110 gpm) to 4.6 ℓ/s (73 gpm), or an average of 34 percent. The average well effectiveness was initially 45 percent; that is, 45 percent of the 5.2 ℓ/s (82 gpm) pumped was diverted from the mine inflow. The well-effectiveness

increased to 55 percent near the end of the test. From these data it was projected that well-effectiveness would increase to about 80 percent after 120 days of pumping with no recharge. Therefore, dewatering well-effectiveness for full scale mine dewatering was estimated conservatively to range between 50 and 80 percent for this mine.

6. The actual range of costs experienced by various mine operators for treating and disposing of approximately the same quality and quantity of water is very difficult to define and compare accurately because of differences in accounting practices used by the mine owners. However, some useful comparisons can be made, as long as their limitations are recognized, and it is apparent that the range of costs for any one element of treatment and disposal can be great. For example, the cost of water treatment alone can vary from less than 10¢/1,000 gal to more than $1.00/1,000 gal, depending on the quality of water to be treated and partly on the water treatment plant capacity or flow rate.

For the study mine, costs for water collection and treatment were computed from company records. It was found that the total capital investment for water collection was $432,000, whereas that for water treatment was $430,000. The annual operating cost for water collection and treatment is approximately $739,000. Of this total, approximately $530,000 is collection, and $209,000 is for treatment. Of the $530,000, approximately 77 percent goes for water collection and transfer costs, and pumping to the surface accounts for 23 percent. Based on an average flow rate of 3.1 million gpd the annual operating costs presented above convert to approximately 46.8¢/1,000 gal for collection and 18.5¢/1,000 gal for treatment. Major cost contributors to the $530,000 annual operating costs are: Power, 37 percent; labor, 36 percent; material, 19 percent; other charges 8 percent. Methods of determining the above figures are presented in Section 9.

The computed values of treatment plant capital cost ($430,000) and annual operating expenses for treatment ($209,000) compare reasonably well with figures from published sources for other plants of similar capacity treating water with similar flow rates and acidity levels.

7. Individually pumped wells constructed from the surface, as used in this study, do not appear to be cost-effective to control water quality at the Lancashire No. 20 mine unless the average well yield can be increased three to four times the 1.9 ℓ/s (30 gpm) used in the analyses. The cost of well dewatering at this mine appears to be, on the average, at least twice as great as present water removal and treatment costs. If the acidity of the mine water were higher (in the range of 500 to 1500 mg/ℓ) this dewatering system would appear attractive. Also, if the coal seam were less than about 45 m (150 ft) deep, it appears that dewatering would be cost-effective using individually pumped wells. In the cost analysis, rather conservative well yields were used based on the assumption that it would be difficult to locate wells along fracture zones, where optimum yields could be obtained. If fracture zones can be located accurately on the surface and penetrated with wells, then it is quite possible that dewatering with this type of system could be cost effective. An average well yield of 1.9 ℓ/s (30 gpm) was assumed in this analysis but, with wells located only in the more intensely

fractured rocks, yields could average as high as three to four times this amount. If an average well yield of 5.7 to 7.6 ℓ/s (90 to 120 gpm) could be obtained, the system of individually pumped wells would be cost-effective.

8. The individually pumped surface wells used in this study constitute the only dewatering system that can be used completely in advance of mining. However, this is the most costly method and requires the most surface facilities. Wells drilled from the surface to intersect mine openings and drained by gravity into the mine are less costly and do not require permanent surface facilities except for discharge points, but they are limited as to location and they do require collection of flows into the mine and pumpage to the surface. Angled drain holes drilled from the mine openings and drained by gravity into the mine appear to be the least expensive option and are potentially the most effective means of mine dewatering where they can be used. They can be more easily varied to meet unanticipated underground conditions, and the system can be constructed by mine crews using conventional underground drilling equipment. This system can be used to dewater to a limited degree in advance of mining and depth of the coal seam has little influence on cost-effectiveness. Calculated costs for these alternative systems (presented in Section 9) suggest that dewatering by means of angled drain holes would be at least marginally cost-effective at the study mine and that this system is worth further consideration.

9. Cost-effectiveness analyses did not consider indirect benefits of dewatering such as reduction of production losses due to high water inflows and unstable roofs. Such production losses due to poor water control can be much more expensive than the acid drainage problems which the water also creates. Dewatering might additionally be combined with methane removal, as well as other improvements that could enable sharing costs. The cost-effectiveness analysis was not a full cost benefit comparison, which is required to determine the true economic feasibility of mine dewatering.

10. It was noted that water degrades progressively as it passes through the mine (item 3 above), and additional monitoring of water transferred through the mine indicated opportunities for separation of good and poor quality waters. Therefore, an alternative to dewatering for control of AMD would be to collect water underground at the point of emergence and discharge it directly before it becomes degraded. This can be done through bore holes located near the areas of inflow and/or with pick-up points and pipelines through the wet areas in the mine. Poor quality water from sources such as old mine workings can be intercepted and treated separately, reducing the amount of water to be treated.

SECTION 3

RECOMMENDATIONS

1. Better exploration techniques are needed to locate and define the extent of fracture-controlled flow paths. Work has been done and is in progress using high- and low-level imagery. This should be supplemented with surface geophysical techniques such as various resistivity surveys and spontaneous potential streaming surveys. These exploration techniques should be proven with test wells and observation wells and hydraulic characteristics determined from pump tests.

2. A pilot dewatering study should be developed for drain holes drilled from the mine openings to develop better cost- and well-effectiveness data. This study could probably be best conducted by the mine operator with specialized help where needed.

3. Water quality sampling within a cross-section of active coal mines should be conducted to determine the opportunities available for prevention of water quality degradation as water is transferred through the mines. Also, a demonstration project could be performed to determine the cost-effectiveness of separating and selectively treating or discharging waters from various areas of a mine.

4. Exploration programs for coal mine development should include geohydrologic data collection. Expensive exploration holes should serve the dual purpose of providing groundwater data as well as information on coal seams. Natural groundwater can be sampled and analyzed for water quality. Holes should be cased to bedrock and preserved as water level monitoring wells. Water level contours can provide valuable information on the preferred flow paths typical of fracture-dominated flow systems.

5. A study is needed to identify the benefits of mine dewatering in addition to water quality control and prevention of mine flooding. Anticipation or prevention of large, sudden groundwater inflows could prevent losses of production time. The improvement in roof stability that could be expected from dewatering in advance of mining would have direct beneficial effects on such problems as lost production time and safety hazards. Potentially, such improvements in operating conditions could bring much larger economic benefits than would control of AMD.

SECTION 4

BACKGROUND

Various investigators[2] have studied the use of well systems to dewater underground coal mines. These investigators have identified most of the technical problems associated with well dewatering systems, and the equipment and technology for construction and pumping are readily available. In practice, however, there is little or no experience of well dewatering of underground coal mines (except for shaft dewatering) that is available to be drawn upon, and the well dewatering systems that have been published are idealized and not necessarily compatible with field conditions. It is questionable whether mathematical techniques are useful in designing well systems for an entire mine where ground-flow is dominated by fracture permeability, which usually results in preferred and erratic flow paths. If the individual flow paths can be identified and their hydraulic characteristics determined, it may be possible to model groundwater conditions mathematically. However, from a practical standpoint, developing these parameters with existing technology presents a very difficult and costly problem. Judging from the experience gained during this study, the condition of fracture-dominated flow seems to be the rule rather than the exception in wet mines of the eastern coal regions.

GENERAL HYDROGEOLOGIC CONDITIONS

In the eastern bituminous coal region, coal seams are typically associated with a series of sandstones and shales (including siltstones, claystones, etc.) with some limestones. Bedding tends to be tabular with gradual pinching out, except for sandstone channels which are lenticular in cross-section. Bedding is generally nearly flat-lying or has gentle dips associated with broad folds or regional tilting. Low-displacement faults are common in many areas, and jointing is variable but often associated with axes of gentle anticlines and synclines. Despite the relatively uniform geology throughout the region, it is dangerous to make generalizations with regard to groundwater conditions.

In the absence of fracturing, the sandstone, limestones, and coal seams act as aquifers because their permeability is relatively high compared to that of the shales, which act as confining beds. Technically, if this solely were the case, it would be rather easy to determine aquifer constants from pump tests and mathematically to predict groundwater response to imposed situations, i.e., dewatering wells. However, fracturing due to jointing and faulting cuts across bedding planes, locally destroying the confining effect of relatively impermeable beds, and bedding plane fractures or partings can create relatively high permeabilities in shales. In zones of relatively intense fracturing, permeabilities are often several orders of magnitude

higher than in unfractured rocks, resulting in fracture-dominated flow along fracture planes. This preferred flow creates extremely difficult problems in predicting groundwater movement and response to induced drainage.

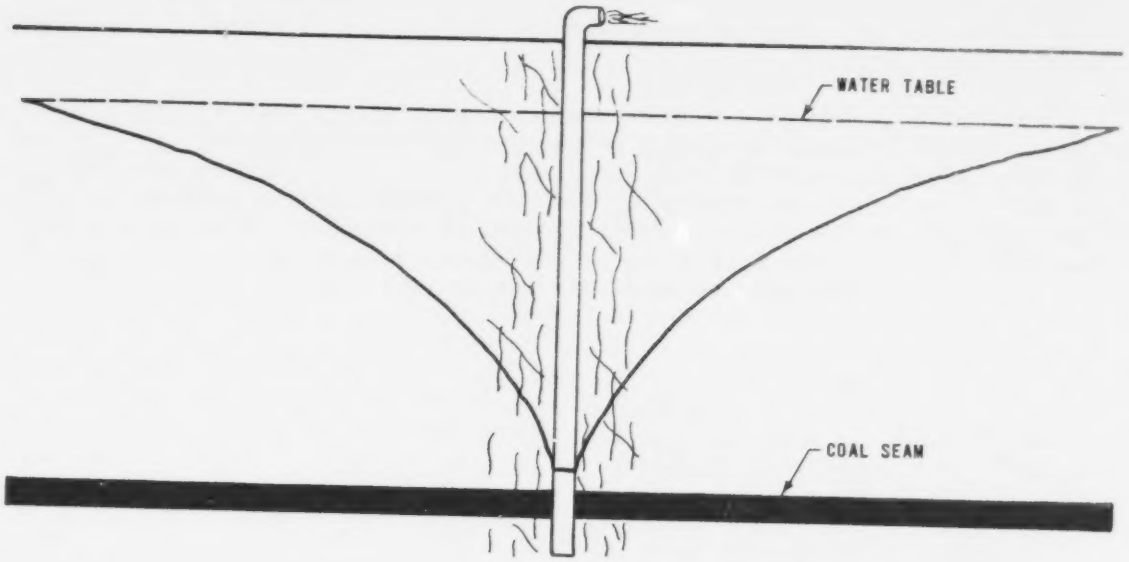
Among the mines inspected during this study, fracture-dominated flow, which tends to localize inflows, appeared to be prevalent in all cases. A critical step in coping with this type of flow system is to develop techniques to locate and define the extent of fracture zones and to develop practical testing methods to determine their hydraulic properties. Considerable work has been done with aerial photographs to locate these zones and further work is in progress. Even if the hydraulic properties of fracture zones remain unpredictable, definition of these zones will make mine dewatering from wells much more practical.

Even in fracture-dominated systems, sandstones, limestones, and coal seams still act as secondary aquifers and probably provide most of the groundwater storage space. If the intersecting fracture zone is drained by a mine opening or by wells, water is released from storage. However, the geometry of the potentiometric gradients becomes complicated because the cone of depression is elongated along the fracture zone (as shown on Figure 1) and the driving forces exerted on the secondary aquifers inducing their drainage is also distorted. Also, drawdowns within the fracture zone may be well below the base of secondary aquifers and may therefore result in variations in discharge, as well as in head, with time. Further, the individual fracture planes in a zone are probably interconnected erratically, so that groundwater response in one fracture plane may be greatly different from the response of a parallel fracture plane a few feet away.

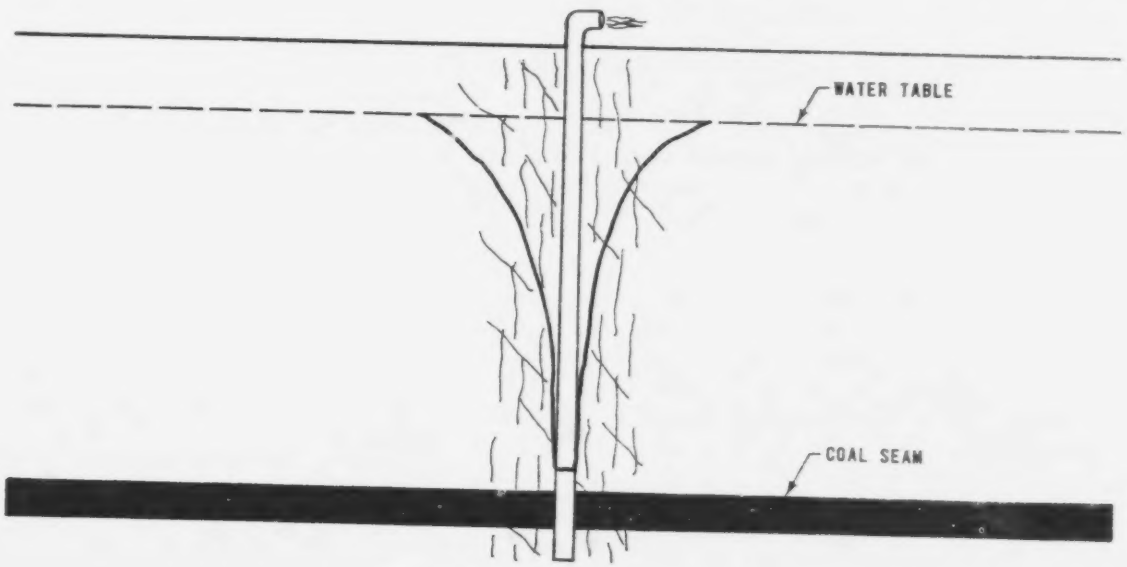
Discussion with other investigators and review of literature indicate that there are areas in the region where fracture-dominated permeability does not control groundwater transmissions. Sandstones, and apparently sandstone channel deposits in particular, can have a wide range in intergranular permeability perhaps ranging from 10^{-8} to over 10^{-3} cm/s. Limestone beds also can be highly permeable if solution cavities are present, and these are probably the second most common source of high mine inflows after fracture zones. (Alluvial and glacial deposits in contact with mine openings are considered a special case.) As indicated previously, these rocks can be treated readily with conventional hydraulic relationships once the hydraulic parameters and extent and thickness are determined through exploration and testing.

HYDROLOGIC EFFECTS OF UNDERGROUND MINING

Under natural conditions, the surface of the zone of saturation tends to conform in a subdued manner to the topography and surface drainage systems. Variations occur, depending on the presence of geologic structural features such as fracture zones and folding or tilting of beds. The variation tends to become more pronounced with depth, where geologic structure provides the primary control. An underground coal mine, as it grows, disrupts natural groundwater flow patterns by acting as a large drain. The flow of groundwater into the mine results in lowering of potentiometric levels above and around the mine. Overlying and underlying aquifers will be affected to varying



1. DRAWDOWN CONE PARALLEL TO FRACTURE ZONE.



FRACTURE ZONE

2. DRAWDOWN CONE ACROSS FRACTURE ZONE.

Figure 1. Schematic asymmetry of drawdowns along a fracture zone.

degrees depending on the leakage characteristics of confining layers or boundaries and depending on the extent to which aquifers are interconnected by fracture zones.

Drainage to the mine is increased by fracturing related to subsidence, which differs somewhat according to the mining method employed. This effect may not be noticeable with room and pillar mining until pillars are removed or until long after the mine is abandoned. Longwall mining methods let the roof collapse in the longwall panel as coal is extracted. With both mining methods subsidence and fracturing may occur to the surface and the result is the development of a new hydrogeologic environment over the mine. This is more apparent with longwall mining because the induced fracturing sometimes results in large increases in mine inflow until most of the stored water intercepted by the new fractures is drained. With both mining methods, inflows of groundwater increase with expansion of the mined area. It is generally believed that the increase during the active mine life appears to be generally greater with the longwall method, although this belief has not been substantiated.

Drainage into mines can deplete overlying aquifers and, of course, affect wells and springs, and stream flow also can be intercepted. Often the heaviest mine inflows are beneath streams, because stream courses are often structurally controlled by synclines, fractures etc., and the streams can provide more recharge to fractures than can water stored in secondary aquifers. Limited observations during this study support this reasoning.

Groundwaters entering active underground coal mines may be of excellent quality or they may be degraded to various degrees by natural processes or by migration through adjacent or overlying abandoned mines. Analyses of uncontaminated groundwaters associated with coal seams in Illinois and Indiana[3] indicated pH values of 7.0 or above and total dissolved solids concentrations ranging from approximately 1000 to 60 000 mg/ℓ. In the Appalachian region the quality of uncontaminated groundwater is generally much better, though there is a general increase in dissolved solids with depth. A representative list of analyses showing ranges of natural water quality for this region was not found, but experience during the study suggests that total dissolved solids for mines 183 m (600 ft) deep or less should often be less than 500 mg/ℓ, with a near-neutral pH. This better quality is probably due to the fact that water transmitted through fracture systems circulates more rapidly, tending to flush more mineralized waters or to preclude their formation.

PROBLEMS ASSOCIATED WITH MINE WATER

The problematic effects of AMD have been extensively described by others and will not be discussed further here. Likewise, the techniques of treating AMD are well known by the industry and readily available in published literature. Neutralization is the most common method of treating AMD.

Most of the direct costs of handling and treating the mine water itself can be readily identified. However, the indirect costs of any water-related

problems are difficult to separate from operating costs. Loofbourow and Brittain[4] outline the problems that are effects of wet conditions in mines:

1. Direct Effects

   a. Cost.
   b. Failure to handle inflow may interrupt production and damage the mine, perhaps beyond recovery, perhaps with loss of life.

2. Indirect Effects in Mine

   a. Freezing water in shafts in cold areas.
   b. Reduced efficiency of crews and equipment.
   c. Added maintenance of equipment, tires, cost or hazard of using electricity.
   d. Washed ground.
   e. Flows of hot water heat and humidify ventilating air.
   f. Dissolved gas carried into some mines.
   g. Reduced stability of rock walls (and roofs).
   h. Increased work on accessways and ditches.
   i. Interferes with certain explosives.
   j. Makes some products unacceptable.
   k. Scale in pipes and pumps.

3. Indirect Effects Outside Mine

   a. Increased shipping, treatment, handling costs.
   b. May pollute surface water (and groundwater).
   c. Drawdown may take water from wells and lower quality of water remaining, or improve it.
   d. Drawdown may cause surface subsidence.
   e. Drawdown may affect surrounding water users.

Many of the problems cited here relate to the simple presence of water in the mine, not directly to the quality of the water. However, some or all of these problems may be positively affected by a solution of the water quality problem that is based on the prevention of water entering the mine in the first place. Some of these problems can have much greater impact on mining costs than that of routine water handling and treatment--for example, interruption of production due to large, sudden inflows or to unstable roofs. Mine dewatering has the potential of reducing the magnitude of such problems and the associated costs. However, these problems vary greatly from mine to mine, as well as varying within a mine from one area to another and from one time to another. The true overall cost benefits of dewatering have to be evaluated on a case-by-case basis. This is best done by the mine engineering staff, with specialized help were needed, because they are most familiar with the operational constraints and conditions of their mine.

CURRENT MINE WATER CONTROL

In shaft and slope mines, water inflows are controlled usually by a collection system of free drainage, ditches, sumps, and pipelines used to transfer water (by gravity flow or pumping) to stations from which it is pumped to the surface. The amount of pumpage used to transfer water to the final discharge point varies greatly with the gradients in the mine and the amount of water handled. This system is flexible in that the collection and transfer stations are expanded as the mine grows and water inflows increase. At the working faces, water is collected in small sumps either by gravity drainage or by small, portable sludge pumps. From the initial sump or collection point, water is transferred through a series of stations that collect increasing volumes of water until the primary discharge station is reached where the mine water is pumped to the surface.

The capacity of the system must be large enough to accommodate large, unanticipated inflows. This is one of the basic weaknesses of the system in that the capacity has to be greater than normally used, or mine production is badly interrupted when large flows are encountered. This is especially true where fracture-dominated flows are encountered. As fracture zones are encountered or as new fractures are created by mining, flows of ten to several hundred liters per second can suddenly enter the mine. Further, these flows may peak and subside to a fairly constant rate or remain high over long periods of time, depending on the recharge source and the volume of groundwater in storage. This places a continuing but erratic strain on the mine water handling system. As a result, the system has to be changed or reestablished periodically and both production losses and over-capacity have to be accepted to some degree. Dewatering in advance of mining or in conjunction with mine advance can help stabilize this situation.

From a water quality standpoint, the current water collection and transfer methods provide opportunities for degradation in water quality by contact with decomposed iron disulfides. This probably occurs mostly on the mine floor, in open ditches, and in open, unlined sumps. As will be demonstrated later in this report, degradation often occurs gradually as mine water is transferred from point to point before reaching the surface. The relatively concentrated inflows might not change significantly in quality if they were collected in the immediate inflow area and transferred from sump to sump by pipeline, even if the sumps were unlined and open. In contrast, widely distributed drips and seeps that are allowed to accumulate slowly on the mine floor and are collected by sludge sumps probably become acid relatively fast. If this widespread inflow is mixed with concentrated inflow that far exceeds it in volume, degradation may not be significant if the water is transferred to the surface rapidly. It is therefore apparent that opportunities probably exist for water quality control underground with current mine water handling systems that could at least reduce treatment costs. This might be accomplished by collecting water at transfer stations close to the source of inflow and pumping either directly to the surface (through boreholes) for discharge to receiving streams, or through a closed system to prevent further exposure to acid-forming minerals.

SECTION 5

SITE SELECTION

A major task of the mine dewatering study was to select an active coal mine in the eastern coal mining region that would be suitable for a pilot-scale dewatering operation. It was decided during project planning to limit the search generally to the region north of the Ohio River in Illinois, Indiana, and Ohio, but to include northern West Virginia and western Pennsylvania. This area, it was reasoned, would provide a good cross-section of the eastern bituminous coal mining region. Also, this area would be more convenient and would therefore facilitate field work by the study team members based with the Ann Arbor, Michigan firms Environmental Technology Control Corporation (ENCOTEC) and Chase-Mogdis, Inc. (C-M). The search team organized by W. A. Wahler & Associates consisted of a hydrogeologist and a mining engineer (both Wahler personnel), a sanitary engineer (ENCOTEC), and an economist (C-M). This team established site selection criteria based on study objectives, schedule, and budget, and divided the selection process into three phases. The three phases consisted of a literature search, a telephone survey, and site visits. Through a process of elimination, this three-phase search resulted in selection of the Lancashire No. 20 Mine, located near Carrolltown, Pennsylvania and owned by the Barnes and Tucker Company.

SELECTION CRITERIA

The search team prepared lists of criteria to use as general guidelines in comparing mine sites. These criteria were flexible because it was realized from the outset that an ideal site probably would not be found. However, minimum conditions had to be met to result in a successful program. The conditions listed below were considered as essential criteria:

1. The mine containing the test site must be an active coal mine in the eastern coal mining region.

2. The mine operator must be cooperative and interested in the program.

3. The mine test area must be overlain by saturated rocks with significant inflows of groundwater into the mine.

4. Mine inflows must be relatively good quality water, but must degrade in quality in the mine, and require treatment before discharge to receiving streams.

5. The test area in the mine must be capable of isolation so that inflows can be measured and sampled separately from other mine water.

6. The land surface over the test site must be accessible to drilling and test pumping equipment.

The study was directed toward the active portion of coal mines, although in many cases it was found that the major sources of water quality problems were adjacent abandoned underground workings. The degradation of ground water in abandoned mines prior to its entry into an active mine is a problem considered to be beyond the scope of this study since it requires a different approach. Inflows of both degraded and non-degraded waters occur (as is the case in the Lancashire No. 20 Mine) but they must be identified and handled separately.

The cooperation of the mine operator was a key site selection criterion. The operator had to be willing to be subjected to a number of inconveniences in supplying information, including operational and capital cost data, and in providing access to the mine for underground monitoring activities. Without this cooperation, and in effect, direct participation and interest in the results of the study, performance of the program would have been impossible.

Saturation of rocks overlying the mine throughout the year was essential for several reasons. Only saturated groundwater flow can be intercepted effectively with dewatering wells. Inflows into the mine must be large enough to create an impact on mining operations by requiring handling and treatment. Groundwater in storage above and adjacent to the mine had to be sufficient to provide a fairly constant inflow and dampen the effects of recharge events due to storms.

Groundwater inflows to the mine must be of reasonably good quality and must require relatively little or no treatment for discharge into receiving streams, but after entry into the mine must become degraded to the extent of requiring treatment for discharge. This criterion was necessary to provide a contrast for comparison of costs between existing methods of handling mine water and interception of mine inflows by dewatering.

The capability of isolating a test area of the mine was important because it was not expected that a significant impact could be made on the total inflow to the mine, or even a major part of it, with a test operation of limited size. The test area must be isolated so that flows could be monitored for flow rate and sampled for water quality.

The last essential criterion was that the land surface over the mine test area must be accessible to drilling equipment. It was believed that the only practical way to develop test data would be to drill wells from the surface and intercept groundwater inflows by pumping. Land ownership, access, cultural features, and topography were all matters of concern.

Another major consideration was the depth of the mine at the test area. The optimum depth range for construction and operation of dewatering wells was believed to be 45 to 120 m (150 to 400 ft). It was felt that shallower test areas might be subject to fluctuations of inflow and water levels due to recharge from storms. Deeper wells would be much more difficult and costly to construct and pump.

Other factors that were of secondary importance, but that were considered, included coal seam thickness; mining method; and the availability of hydrogeologic, water quality, and cost data. The coal seam thickness and mining method were considered mainly with respect to working room and accessibility of the underground test area. The availability of information would influence both the effort expended on data collection and exploration and also the quality of the work product.

SELECTION PROCESS

The selection process was divided into three phases consisting of a literature search, a telephone survey, and site visits. The Keystone Index of coal mines and state lists of mines were used as the primary reference to develop an initial candidate list. The mines selected from the Keystone Index were limited geographically to Illinois, Indiana, Ohio, Pennslyvania and northern West Virginia. Further, the mines were limited to slope or shaft mines because it was reasoned that drift mines would be more likely to have more seasonal water inflow problems and would tend to be located in more rugged topography. Also, mines deeper than 120 to 150 m (400 to 500 ft) were avoided. Review of other literature was not fruitful in identifying potential candidate sites, but it did provide general information on the magnitude and type of mine water problems.

With the information available, a telephone survey was performed. Both mine operators and state and federal agencies were contacted throughout the survey area. This survey provided clues to general mining areas and to specific mines or mine operators where mine water was a problem. It soon became apparent that the most suitable sites were probably in southeastern Ohio and western Pennsylvania. The candidate list was narrowed to eleven mines as a result of this survey, and appointments were made with mine operators to discuss the program and with state and federal agencies having localized knowledge of specific mine water conditions.

Discussions with mine operators reduced the candidate list to nine mines in Ohio and Pennsylvania. Formal permission was then requested both to inspect site conditions and to use the mine in question for the pilot dewatering study if it proved to be the most suitable site. Negative responses were received from the owners of four mines, which reduced the number of candidate mines to five, all in western Pennsylvania.

During the spring and early summer of 1976, the five candidate mines were visited and inspected in detail. The mines visited were the Vesta No. 4, Shannopin No. 2, and Lancashire No. 24D mines owned by the Jones & Laughlin Steel Corporation; the Lancashire No. 20 mine owned by the Barnes & Tucker Company; and the Rushton mine owned by the Rushton Mining Company. The mine operators were very helpful and cooperative at all five mines. Each site was inspected at least once, and additional surveys were performed at three mines before a final choice was made. After the first inspection, the Vesta No. 4 and the Shannopin No. 2 mines were eliminated for technical reasons. Additional visits had to be made at the three remaining sites to collect water samples for analysis and other detailed data for the final comparison. The suitability of each mine for the proposed study is summarized in the following description.

Vesta No. 4 Mine

The Vesta No. 4 mine located near California, Pennsylvania is a shaft mine using room and pillar extraction of coal from the Pittsburgh seam of the Monongahela Group. This mine was eliminated as a test site because of the suspected nature of the water inflows. The coal seam crops out in a stream channel; inaccessible mine workings at a shallow depth may have partially subsided, permitting stream inflow through fractures. The inflow was estimated to be over 126 ℓ/s (2,000 gpm) where it could be monitored. The chances of intercepting a significant part of this inflow with wells was uncertain at best. The risk of developing inconclusive data appeared to be too high.

Shannopin No. 2 Mine

The Shannopin No. 2 mine, located near the Vesta No. 4 mine, is a shaft and drift mine using room and pillar extraction of coal from the Pittsburgh seam. This mine was eliminated because the mine drainage water does not require treatment for discharge to receiving streams. Hydrogeologic conditions appeared amenable to dewatering; however, it was obvious that the cost-effectiveness of dewatering would be negative from the standpoint of water quality because the water did not require treatment.

Lancashire No. 24D Mine

The 24D is classified as a slope mine using a longwall method of extraction of coal from the Lower Freeport ("D") seam, which is about 1.0 to 1.2 m (40 to 48 in.) thick. This mine, located near Barnesboro, Pennsylvania, was eliminated because the water pumped from the mine does not require treatment before it is discharged into the receiving stream. Initially it was thought that the quality of the mine water was degraded enough to consider the site. However, after the samples had been collected and analyzed, the results indicated otherwise, as shown on Table 1. Otherwise, the mine met all of the requirements for the test site. The coal seam is 70 to 90 m (250 to 300 ft) beneath the surface and the major inflows occur from a combination of fractures and a sandstone channel along a syncline.

TABLE 1. WATER QUALITY - LANCASHIRE NO. 24D MINE

Sample	pH	Total Alkalinity mg/ℓ as $CaCO_3$	Total Acidity mg/ℓ as $CaCO_3$	Sulfate mg/ℓ	Total Iron mg/ℓ
Mine Inflow	7.45	138	--	110	0.038
Mine Discharge	7.92	78	--	210	0.48

Rushton Mine

The Rushton mine located near Philipsburg, Pennsylvania is a shaft mine using room and pillar extraction of coal from the Clarion ("A") seam. This mine was eliminated because of poor quality water inflow to the mine. Initial inspection of this mine indicated that it had most of the prerequisite conditions needed for the test site. It appeared that one heading in a fault block structure with a possible sandstone channel would serve as an ideal test area. It had an added advantage of occurring only 60 to 75 m (200 to 250 ft) beneath the surface. However, the water inflows were sampled and analyses revealed that the water quality was degraded before entering the active mine. The source of this poor quality water is apparently some abandoned workings in the Lower Kittanning ("B") coal seam, which is about 25 m (80 ft) above. Groundwater above the Lower Kittanning seam is of good quality, as shown on Table 2.

TABLE 2. WATER QUALITY - RUSHTON MINE

Sample[a]	pH	Total Alkalinity mg/ℓ as $CaCO_3$	Total Acidity mg/ℓ as $CaCO_3$	Specific Conductance µS/cm	Sulfate mg/ℓ	Total Iron mg/ℓ
R-1	4.43	-	437	1270	1060	180
R-2	4.45	-	909	2100	2040	364
R-3	3.07	-	428	1220	1020	102
R-4	3.04	-	540	1680	1320	163
R-5	6.80	4	-	1500	1150	0.97
R-6	3.67	-	42	270	96	4.02
R-7	6.30	3	-	50	21	0.04

[a]Location of Water Samples:

R-1 Roof, East Main

R-2 Roof, first left North Main

R-3 Mine discharge water before treatment

R-4 Mine discharge water before treatment

R-5 Treated Discharge water

R-6 River water above Rushton Treatment Plant discharge

R-7 Shallow well water above "B" seam

Lancashire No. 20 Mine

The No. 20 mine is located within a few miles of the No. 24D mine. It is also a slope mine utilizing the longwall mining method to extract coal from the Lower Kittaning seam, which is about 1.5 m (60 in.) thick. Table 3 shows that the quality of mine inflow water is very good and, if intercepted, could be discharged directly into a stream without treatment. This table also shows that the quality of the mine water deteriorates as it is transferred through the mine to the F-1 or F-14 sumps, where it is pumped to the surface. The water then requires treatment before discharge to the receiving stream. Several inspections revealed that a small area of the mine with reasonable inflows--6 to 10 ℓ/s (100 to 150 gpm)--could be readily isolated for monitoring of flows and water quality. Mine inflows were transmitted by fractures in a thin shale bed overlying the coal seam. It was thought that a thick sandstone bed overlying the shale would act as an aquifer which could be penetrated by wells to intercept mine inflow. The principal problem with this site was that the test area of the mine is 150 to 170 m (500 to 550 ft) beneath the surface and wells had to be placed on a moderately steep sidehill. Also, surface rights are not owned by the mine, and therefore access rights would have to be acquired from the landowner. However, after comparisons with the other available sites, the Lancashire No. 20 mine was selected as the best alternative for the pilot dewatering operation.

TABLE 3. WATER QUALITY - LANCASHIRE NO. 20 MINE[a]

Sample	pH	Total Alkalinity mg/ℓ as $CaCO_3$	Total Acidity mg/ℓ as $CaCO_3$	Sulfate mg/ℓ	Total Iron mg/ℓ
Mine Inflow	8.04	115	--	41	1.59
Mine Discharge before Treatment					
F-1 Sump	3.03	--	105	880	50
F-14 Sump	6.27	15	--	610	12
Surface Water Above Treatment Plant Discharge	7.42	26	--	44	1.40

[a] The Barnes and Tucker No. 20 Mine Treatment Plant treats the combined F-1 and F-14 flows and by NPDES permit must produce an effluent with a pH between 6.50 and 9.0, with a total iron concentration not to exceed 4.0 mg/ℓ.

DEVELOPMENT OF A PILOT DEWATERING PLAN

In conjunction with final site selection, a program plan was developed for the pilot dewatering operation. The program plan was discussed by the project team and formulated in late July, 1976. The program consisted of the following elements:

- Baseline data collection
- Construction of a test well and observation wells
- Initial pump testing and pilot dewatering system design
- Construction of the dewatering system
- Operation of the pilot dewatering system
- Schedule and costs

The basic plan was to develop wells adjacent to the underground test area that would develop a lateral barrier to mine inflow and reduce heads in the sandstone above the test area. It was decided to maintain a distance of 30 m (100 ft) from the mine face for safety reasons except for one observation well partially penetrating the overlying sandstone. Since the mine was known to have a methane problem, it was judged to be too risky to penetrate the coal seam any closer without disrupting mining operations.

The dewatering plan was predicated on the assumption that the sandstone bed over the coal seam would act as an aquifer. This turned out not to be the case. Fracture permeability dominated groundwater flow at this site and the fractures cut across bedding planes, resulting in unpredictable hydraulic characteristics of the rock units. Instead of being able to design wells based on pump test data from a single well, the dewatering system had to be developed almost on a trial-and-error basis because hydraulic responses could not be predicted for even a few meters. As a result, each dewatering well had to be developed and tested separately, and even these tests turned out to be deceptive.

SECTION 6

SITE CONDITIONS

DESCRIPTION

Mine

The mine selected for study in this investigation, the Barnes and Tucker Company's Lancashire No. 20, is an active mine located near Carrolltown, Cambria County, Pennsylvania (Figure 2). The Lancashire No. 20 is a moderately deep slope mine, and the Lower Kittanning ("B") coal of the Allegheny group is the only seam mined. The coal seam averages 1.5 m (60 in.) in thickness and is located at depths ranging from 150 to 170 m (500 to 550 ft) in the vicinity of the study area. The "B" coal is a metalurgical grade, low sulfur, low volatile coal.

The longwall method of mining is used. Coal is cut with an Anderson Mavor double drum ranging shear, and the roof is supported with a Douty 10-ton shield with self-advancing chocks. Coal is transported by 91 cm (36 in.) and 107 cm (42 in.) conveyor belts that exit from the mine through a slope shaft. Driving of the mains and entries for development of the longwall panel is accomplished with continuous miners. During this latter operation, coal is removed by shuttle cars and unloaded on to the conveyer belts. The mine is normally operated in three shifts and has a daily capacity of 3200 t (3,500 ton) of cleaned coal.

The mine area lies within the drainage basin of the Susquehanna River, whose flow ultimately reaches the Atlantic Ocean. Surface runoff from the part of the mine area selected for the pilot dewatering program drains directly into Laurel Lick Run, which lies downslope from the test area. Laurel Lick Run, with a drainage area of approximately 2331 ha (9 sq mi), flows into Chest Creek (near Bradley Junction), which in turn flows into the West Branch of the Susquehanna River. Laurel Lick Run and Chest Creek are two of the streams in the area which have not been severely polluted, and which support trout and other aquatic life. In the past Chest Creek has been used as a source of municipal water and could be used in this manner in the future. Because of the high quality of water in this watershed, treated mine drainage from the Lancashire No. 20 mine is discharged into the West Branch Susquehanna and not into Laurel Lick Run, which is closer.

Mine Water Conditions

The Lancashire No. 20 mine, along with many other mines in Cambria County, has continually had problems with groundwater inflow. This drainage into the

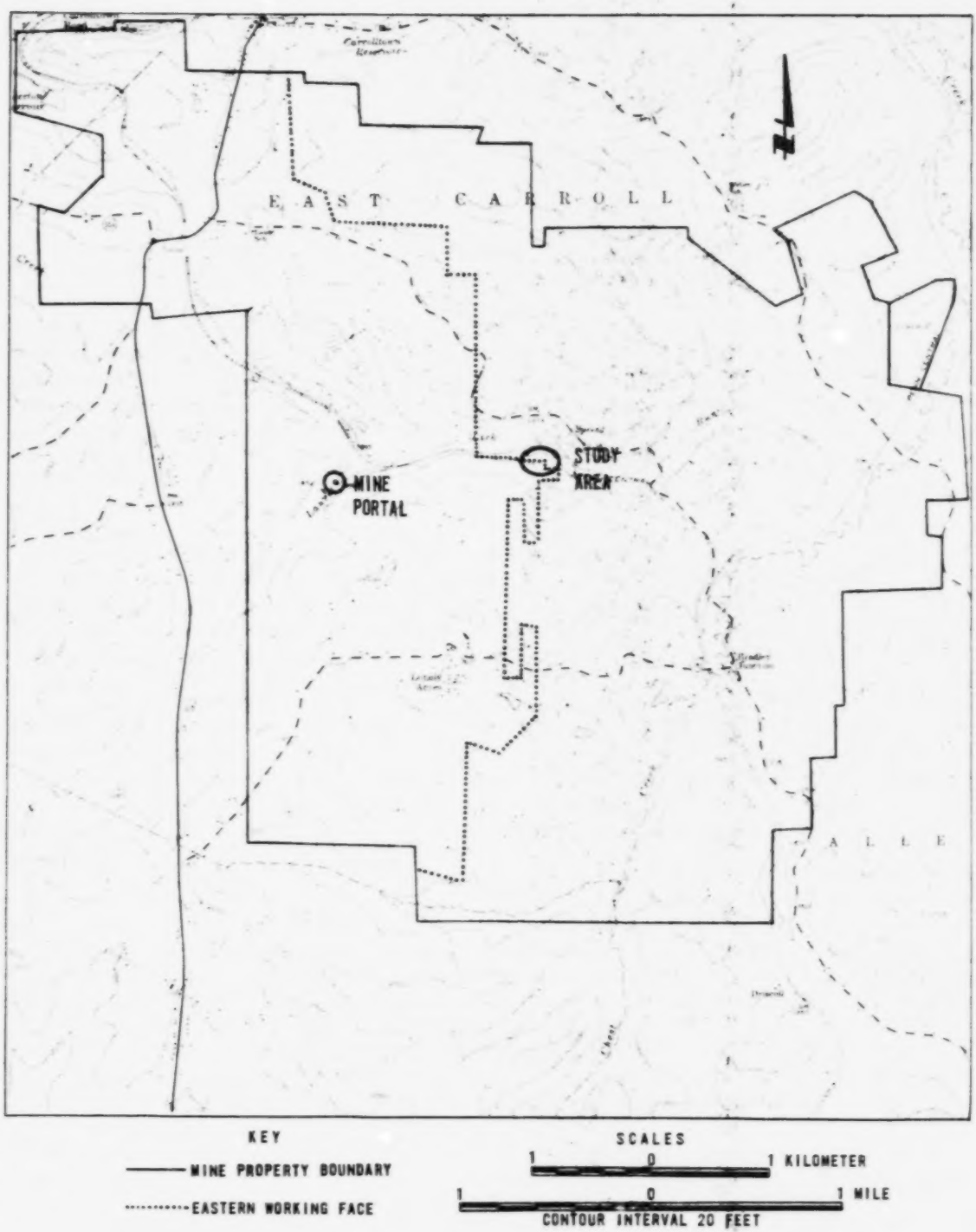


Figure 2. Location map: Lancashire No. 20 mine and pilot well dewatering site.

mine has had a significant impact on working conditions and on the overall cost of mining. To control this inflow, the mine has constructed a water handling system consisting of pumping stations that transfer water through a series of pipelines and sumps.

The source of water inflow to the mine is from groundwater in overlying and surrounding rocks. Water inflows to the mine occur primarily through the roof and exterior faces and water is transmitted predominantly by fractures. There is probably also some inflow through the mine floor, but it appears to be relatively minor in comparison. The mine floor is on an underclay (clay-shale) that restricts upward seepage of water.

Although minor drips and seeps are scattered throughout the mine, the larger inflows are localized, probably along zones of more intense fracturing. Fractures and fracture zones are developed originally by geologic processes and later by fracturing associated with subsidence of rocks overlying the mine openings. The natural fractures are encountered while driving main haulage ways, and while establishing entries and cross cuts during the development of longwall panels for mining. Unstable roofs are often associated with areas of relatively high water inflow.

Inflows into the longwall panels and into abandoned areas of the mine are influenced both by natural fractures and subsidence fractures. The longwall panels are 150 to 170 m (500 to 550 ft) wide and several thousand meters long. Normally, all of the coal is extracted in a continuous operation and the roof support is moved forward with the coal cutting machine, allowing the roof behind the support to collapse. Entries previously driven alongside the panel partially dewater the panel area, but new fractures associated with roof collapse, along with pre-existing fractures, promote more drainage into the mine. New fractures may extend to the surface, resulting in drainage of groundwater that was not tapped previously. These mechanisms can result in inflows of water that are large enough to cause handling problems and that have hydrostatic heads sufficient to affect the caving characteristics of the roof--both of which can affect production. Similarly, additional fracturing can be caused by the slow collapse of abandoned or inactive openings in which roof supports have not been maintained.

It appears that approximately 40 percent of the water enters through abandoned parts of the mine and 60 percent enters directly into the more recent workings. Inflows are affected by seasonal conditions and it is even possible to recognize recharge from individual rainstorms. There is a tendency for high inflows to occur beneath stream channels, which probably tend to follow fracture zones along parts of their course. Wells and springs overlying the mine may be affected or drained completely, especially where subsidence has occurred.

Water Removal and Treatment

Water in the mine is collected and transferred through a series of sumps and conduits and eventually pumped to the surface. Initially water is collected by small sludge pumps or allowed to drain by gravity into sumps. From there it is pumped from sump to sump through pipelines, as shown schematically

on Figure 3. Water is pumped to the surface from the F-1 and F-14 sumps and the pipelines are combined to a single flow at the treatment plant, which has a design capacity of 19 000 m^3/d (5 mgd). Flow to the plant is controlled by pump operators underground, according to water levels in the sumps. Logs are kept for each pump and are used to determine flow rates, based on the pump curves for the two sumps. Figure 4 is a graph of average daily flows to the treatment plant. The pattern of these flows tends to follow the dry and wet periods for this region. The yearly average flow for the period July 1976 through June 1977 is 13 000 m^3/d (3.4 mgd). The data after June 1977 are not reflective of normal flows, due to the flooding on July 19-20 and cleanup operations extending well into August 1977.

At the treatment plant, in a chemical mix tank (see Figure 5), lime in slurry form is added to neutralize the water and to raise the pH to a sufficient level to cause iron precipitation. The lime is added manually, and quantitites are adjusted according to data from daily observations and weekly chemical analysis of effluent. Suspended solids and iron precipitate are then removed in settling ponds, and the effluent from the ponds is piped to a company-owned facility in another watershed for discharge into the West Branch Susquehanna River. Effluent from the treatment plant is monitored weekly for pH, alkalinity/acidity, and total iron. According to data provided by mine personnel, the average levels for the period June 1976 to August 1977 were:

	Average	Standard Deviation
pH	8.7	0.7
Alkalinity (mg/ℓ as $CaCO_3$)	60	20
Total Iron (mg/ℓ)	2.1	1.4

Since the mine owners buy lime in bulk to serve all treatment and processing needs at more than one plant, records of quantities used in individual situations were limited. However, plant personnel made an intensive effort over several weeks to define usage specifically for this treatment plant, and according to the results of this study the average usage was 1360 kg (3,000 lb) of 95 percent CaO (calcium oxide) per day.

Underground Study Area

The underground test site herein referred to as the "Main G study area" was located near the northeasternmost limit of workings along Main G heading, or (as commonly referred to in the mine) the left side of Main G, near section G-14. The study area is bounded by the haulage way on the south; by the edge of underground working on the north and east; and by the G-11 entry on the west. Figure 6 presents a map of this portion of the mine.

This area was selected because it presented relatively high groundwater inflows estimated at 6 to 10 ℓ/s (100 to 150 gpm) and because it could easily be separated from other mine drainage and monitored. Mining in the immediate area had been temporarily halted because of unstable roof conditions and high groundwater inflows, so advancement of the workings would not hamper the progress of the project.

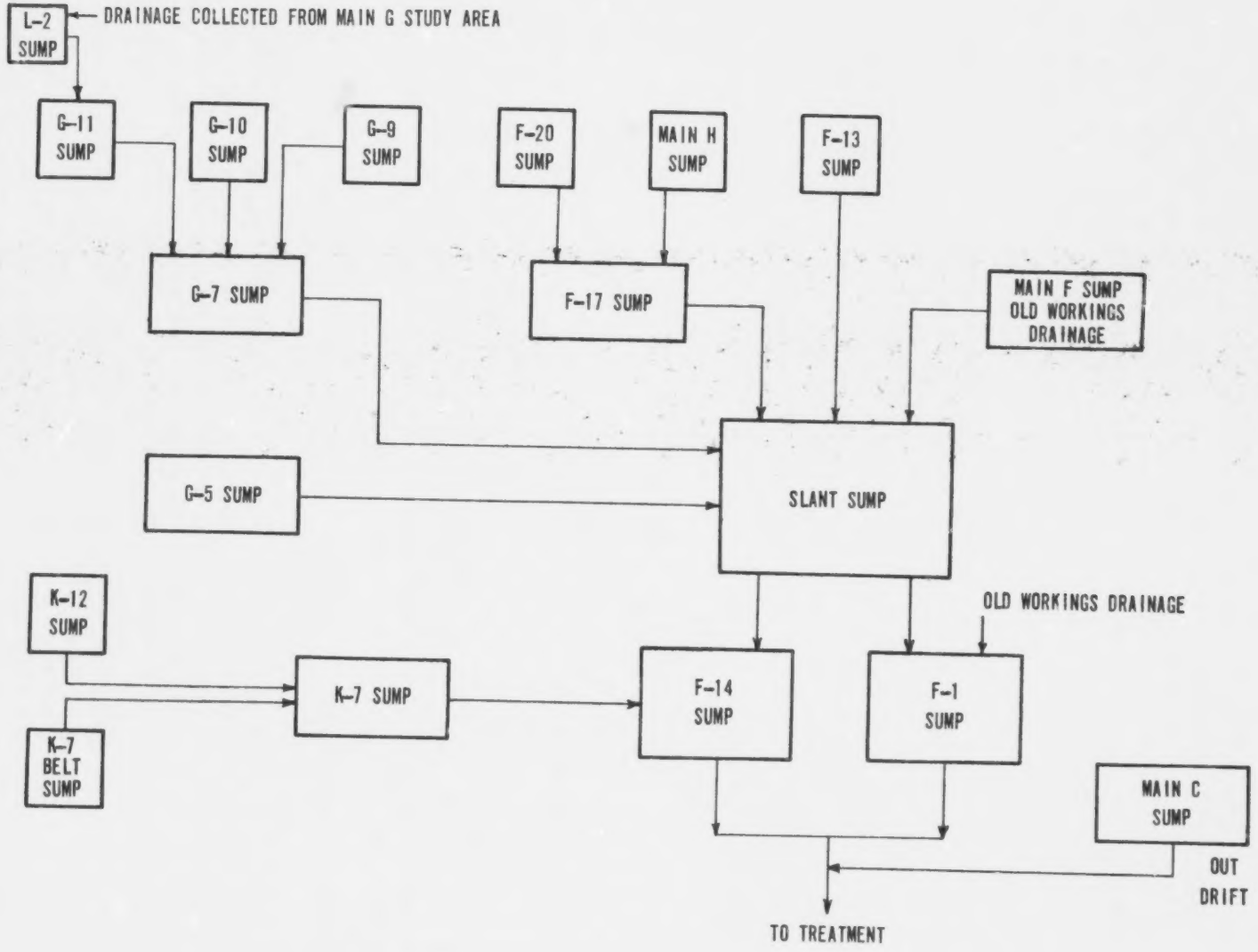


Figure 3. Water transport system.

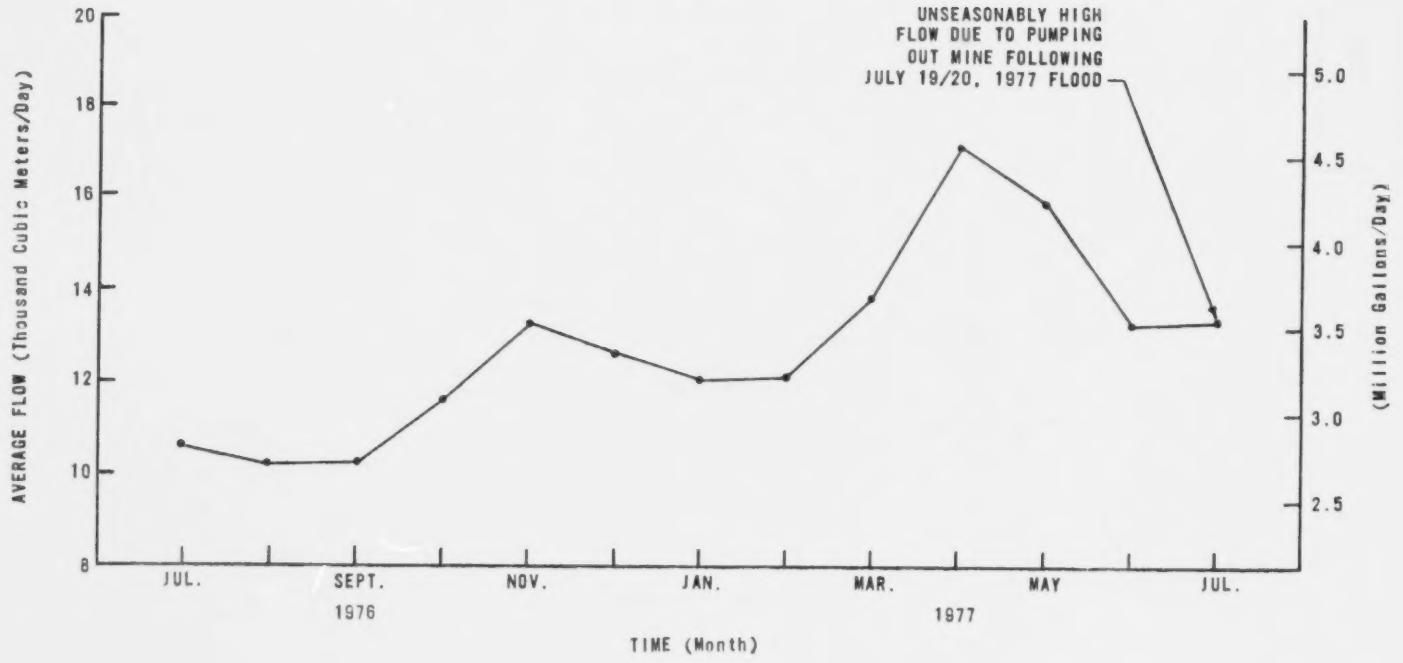


Figure 4. Average daily flows to treatment plant.

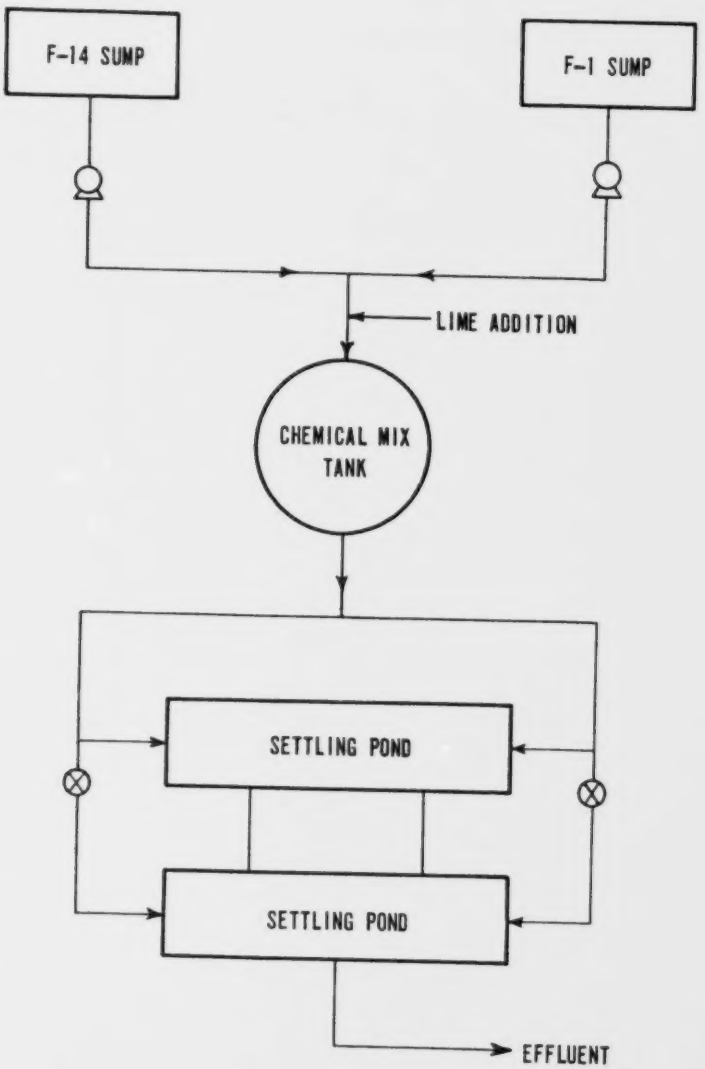


Figure 5. Treatment plant - schematic layout.

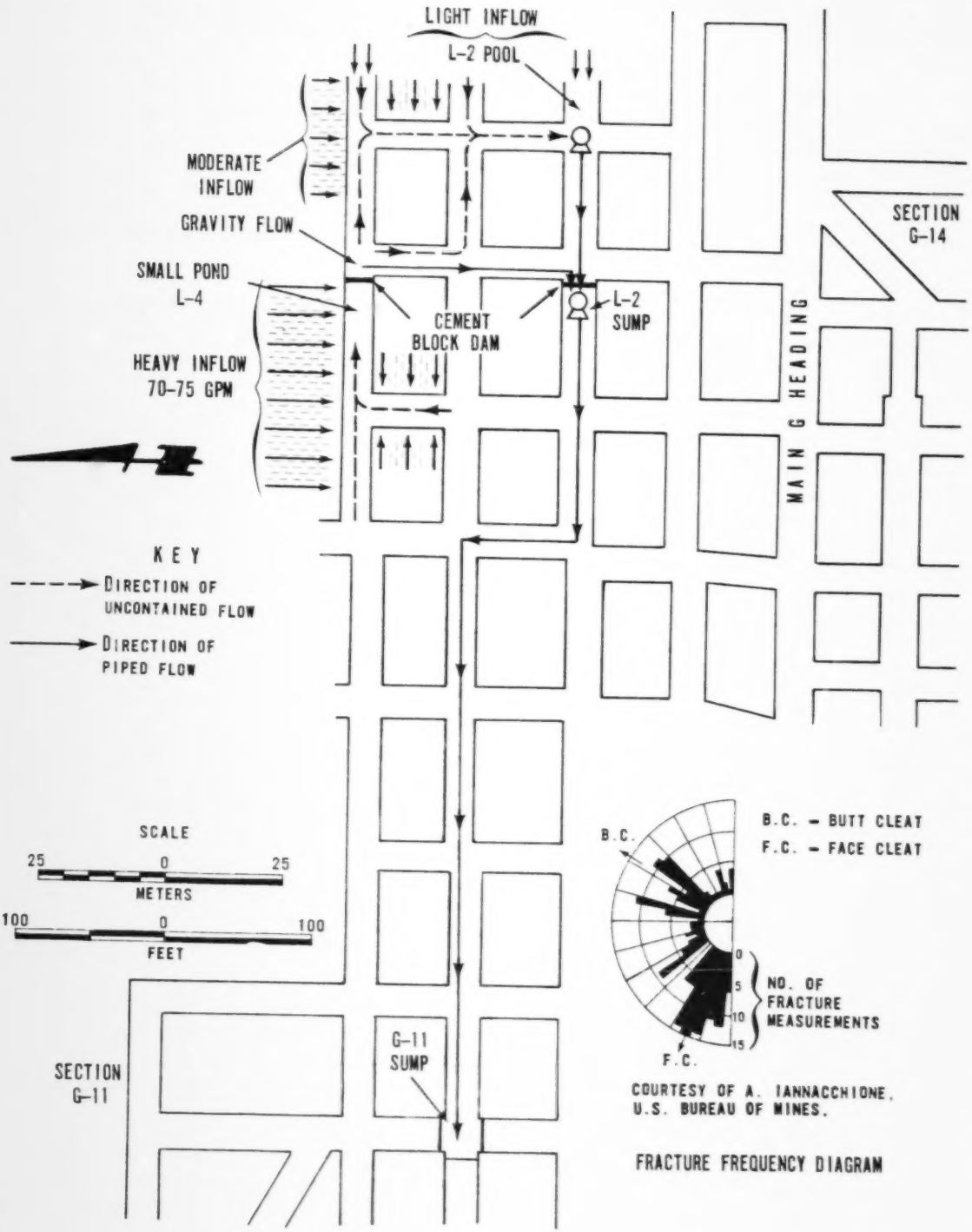


Figure 6. Main G study area.

Water enters the study area of the mine from the roof and from the coal seam. The roof is a gray to dark gray-brown shale-siltstone, which is fractured to intensely fractured. Slickensides and polished surfaces are common on much of the rock, and water drains freely from these fractures. Fractures in the roof rock are common throughout this area of the mine. Water inflow, however, tends to be more localized in areal extent. The shale-siltstone roof is about seven feet thick and is overlain by a massive sandstone. Originally, it was thought that this sandstone was acting as an aquifer and that drainage from it was the source of groundwater inflow to the study area. However, drilling and pump testing indicated that this unit is relatively tight except along fractures.

The area of greatest mine inflow is along the L-4 heading near the northern face (see Figure 6) where most of it is collected behind a cement block dam. Overflow from the L-4 pond and any additional drainage from the left (north) side of Main G heading are collected in a low area of the mine floor, forming a pool located at the east end of the L-2 heading. Prior to establishment of the underground monitoring system, the L-2 pool also collected drainage from the right (south) side of the Main G heading. Water from both the L-4 pond and L-2 pool are then transferred to the L-2 sump, which is formed by a cement block dam and is also located in the L-2 heading. Gravity flow is utilized to drain the L-4 pond, whereas pumping is required to remove water from the L-2 pool. Water collected in the L-2 sump includes all of the mine inflow to the Main G study area. From L-2 sump, water is then pumped to the G-11 sump where it is combined with water from adjacent areas and transferred through the mine and eventually to the surface, as shown schematically on Figure 7.

INVESTIGATIVE ACTIVITIES

The investigative activities performed for this project had a dual purpose: first, to establish the geologic and hydrogeologic conditions for use in the design, construction, and operation of the pilot dewatering program; and second, to develop baseline data on inflows and water quality. The results of the hydrogeologic aspects of the investigation appear later in this section. The pilot dewatering program is presented subsequently in Section 7. The investigative activities included:

- Development and construction of underground monitoring facilities.
- Reconnaissance geologic mapping.
- Drilling and construction of observation and dewatering wells, with logging of drill holes to provide hydrogeologic data.
- Pump testing of dewatering wells.
- Monitoring of underground and surface waters to develop baseline data.

Investigative activities began in September 1976, with the establishment of the underground monitoring facilities and surface geologic mapping. Drilling and pump testing operations followed these and continued into December, when extremely difficult weather conditions slowed down progress. Due to the severity of the weather, field activities had to be temporarily demobilized in late January. Operations resumed in May and all drilling and testing

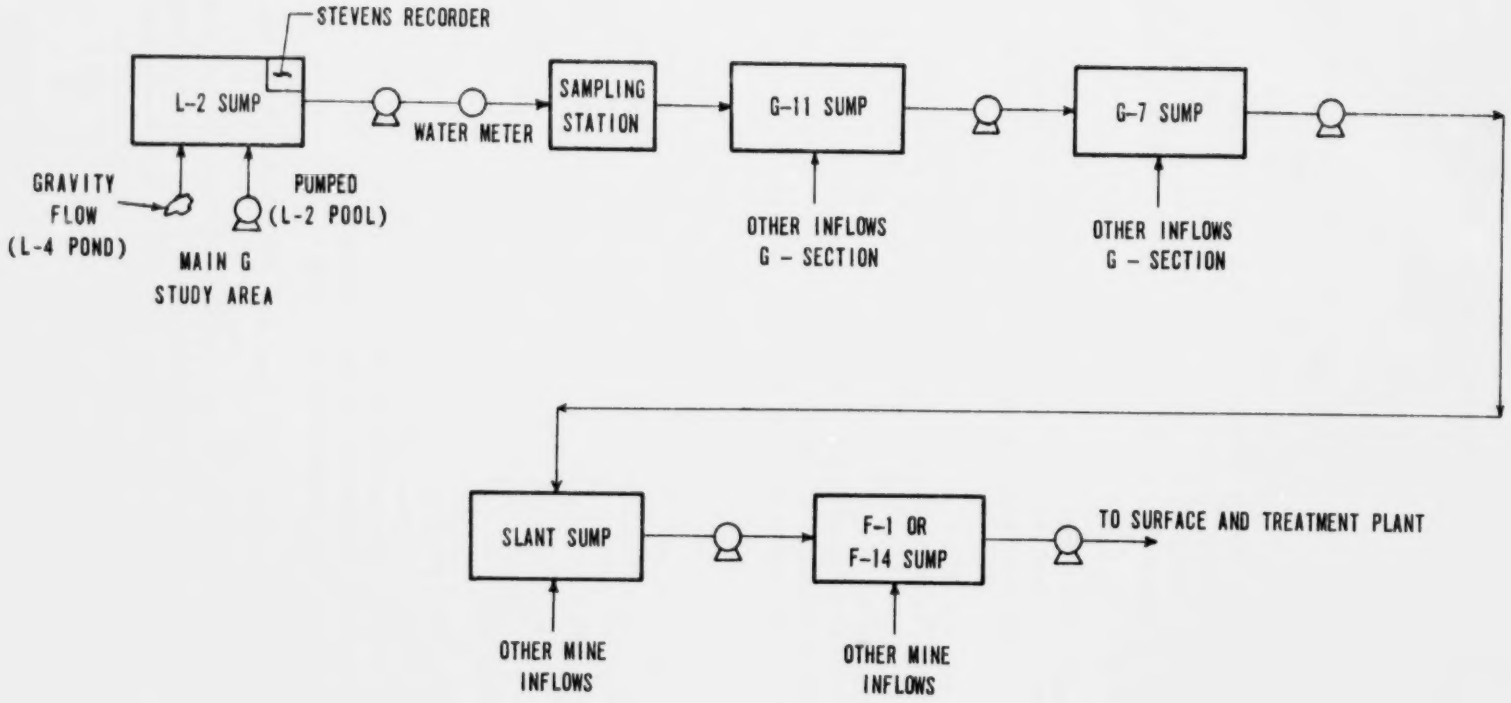


Figure 7. Mine water flow path and monitoring system - Main G study area to treatment plant.

activities were completed in June 1977. Monitoring activities continued at least on a limited scale without a break until pilot dewatering operations were initiated in July 1977.

Underground Monitoring

The underground monitoring facilities were designed to obtain background data on quality and quantity of mine inflow, and to provide a means for detecting changes in these during the pilot-scale dewatering. To monitor mine inflow, it was necessary to isolate the underground study areas from surrounding mine drainage. This was accomplished by diverting all flow from the right (south) side of the Main G heading to the G-11 sump located to the west of the test site. Mine drainage from the study area was then collected in the L-2 sump described previously.

After this diversion was completed, only water inflows from the study area were collected and pumped from the L-2 sump. Figure 7 schematically shows the water flow path out of the mine and the monitoring system established for this study.

Monitoring the rate of mine inflow was accomplished using a 7.8 cm (3 in.) Rockwell flow meter attached to the discharge line of the pump draining the L-2 sump. The water level in the pond was monitored in conjunction with this to record periods of pumping and non-pumping. Water-level monitoring was accomplished using a Stevens water-level recorder. Mine inflow was then calculated using data obtained from both of these measurements.

Samples for water quality analysis were obtained from a sampling box (see Figure 8) built near the end of the discharge line from the L-2 sump to the G-11 sump. Samples were collected using a Protech pressure-operated composite sampler. No rock particle sediments accumulated in the box during the monitoring period. Additional samples for water quality analysis were obtained from inflows from the roof near the L-4 pond. These were "grab" samples collected directly from the mine roof. All monitoring and sampling equipment used was inspected by state and federal mine inspectors prior to its installation in the mine.

The flow meter was monitored and water samples from the Protech sampler were collected by mining personnel. Flow meter readings were taken each shift, or three times daily, during the routine inspection of the pump at the G-11 sump. Twenty-four-hour composite water samples from the sampler were collected only during the pilot dewatering operation.

The Stevens water-level recorder was monitored by personnel from the study team. The water-level recorder was set on an eight-day time cycle, and chart paper was changed weekly. During each mine visit, the underground study area was inspected to insure the continued separation of mine drainage and the continued collection of flow meter readings. Grab samples of inflow near the L-4 pond were also periodically collected during these visits.

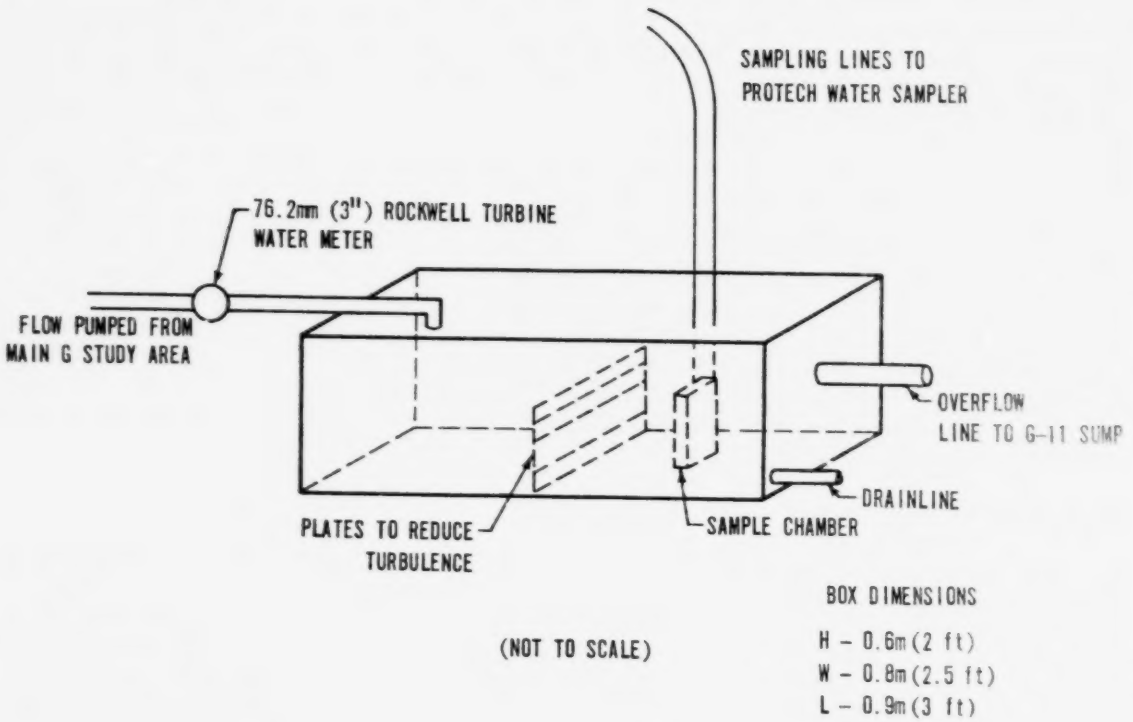


Figure 8. Sampling box at G-11 sump (schematic).

The monitoring system as described was in operation from September 1976 to July 1977. During the night of July 19-20 the mine was flooded as a consequence of surface runoff into the shaft following a severe thunder storm (which flooded Johnstown and other communities in the vicinity), and mining operations were temporarily halted until the mine could be dewatered.

Following storm water removal and cleanup at the mine, the underground water transport scheme was reorganized and isolation of the left side as previously maintained was no longer possible. The new system required temporarily directing all flow from the left side away from L-2 sump and toward L-2 pool. This was accomplished by plugging the gravity line draining L-4 pond and allowing the water to flow by gravity to the L-2 pool, where all inflow from the left side was now collected. (L-2 pool had been previously used to collect overflow from L-4 pool and other additional drainage from the left side which did not flow directly into L-2 sump.) Monitoring and sampling were performed as described earlier; however, both instruments were now located at L-2 sump near the end of the L-2 pool discharge line. No change was noted in the chemistry of the monitored flow as a result of rearranging the monitoring devices. The revised monitoring system and water flow path is shown schematically on Figure 9. This new system had two disadvantages. First, it allowed more lag time between any changes in mine inflow and actual measurements. Secondly, recirculation of L-2 pool water was possible, since overflow from L-2 sump would flow by gravity into L-2 pool. If this occurred, a higher inflow rate would be calculated if corrections were not made. To correct for times of overflow, the Stevens recorder was again monitored at the L-2 sump.

Geologic Mapping

The ground surface overlying the underground test site was geologically mapped in September 1976, prior to the start of exploratory drilling. The purpose of the mapping was to determine whether surface evidence existed for geologic control of mine inflow and also to aid in locating future dewatering wells. Low-elevation aerial photos were reviewed in the performance of this task. Generally the terrain proved not to be amenable to mapping and this part of the investigation contributed only minor information regarding possible well locations. However, it was possible to identify the general geologic features of the site area. The reconnaissance geologic map (Figure 11) is presented with the geology discussion later in this section.

Well Construction and Pump Testing

Well Construction - A total of seven wells was drilled from the surface to obtain subsurface geologic data and for use in the well dewatering program. Well locations are shown on Figure 10, and drill logs are presented at the end of this section on Figures 16-22. All drilling was done with air-rotary rigs, initially with a single compressor Davy rig and later with a larger, dual compressor Schramm rig. Drilling and all subcontracted field work was performed by Toy Drilling of Distant, Pennsylvania.

Initially three wells were drilled. Locations of these wells were determined primarily by areas of high inflow in the underground test site, and

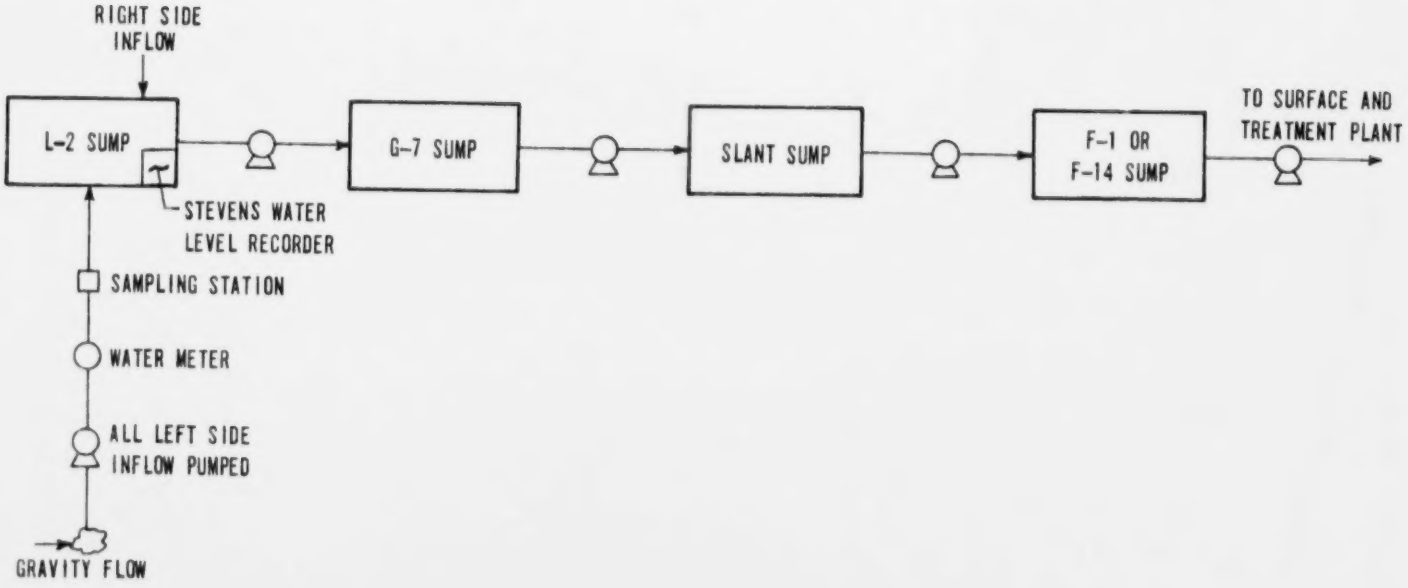


Figure 9. Revised mine water flow path and monitoring system - Main G study area to treatment plant.

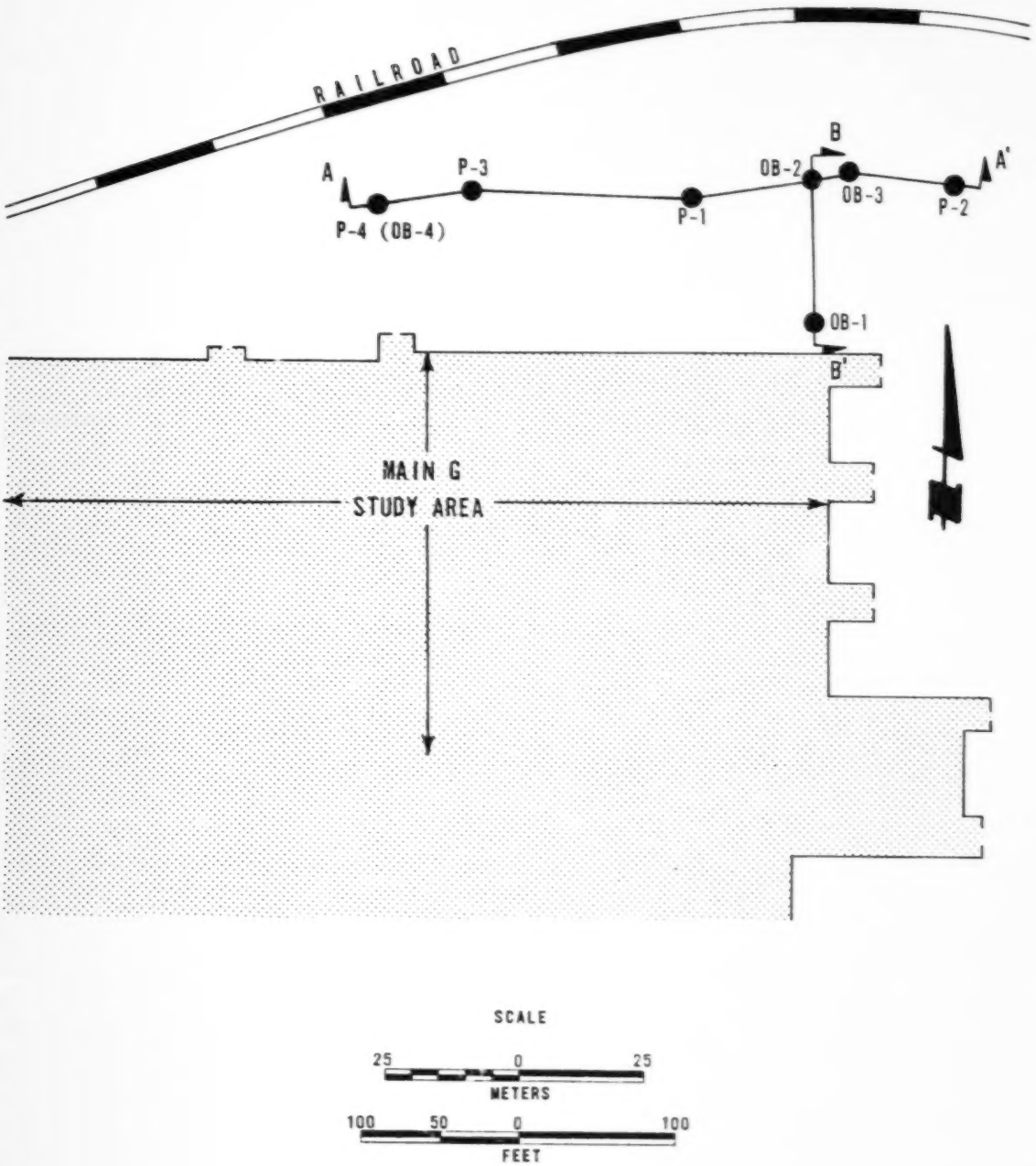


Figure 10. Location map: Dewatering and observation wells.

by surface topography and accessibility. Test well P-1, 20 cm (8 in.) in diameter, was located approximately 30 m (100 ft) horizontally from the outermost mine workings in L-4, and drilled to the base of the "B" coal seam. Two 15 cm (6 in.) diameter observation wells, OB-1 and OB-2, were drilled in conjunction with P-1. These holes were located 9 m (30 ft) and 37 m (120 ft), respectively, from the mine workings. Termination depths were 15 cm (50 ft) above the mine roof for OB-1, and the base of the "B" coal seam for OB-2. During the drilling of each of these wells, drill cuttings were inspected and identified and a drill log prepared. This and comparable data derived from holes drilled later were used to construct a stratigraphic column (Figure 12, present on page 40 with geology discussion). Also, water encountered in and air-lifted from the holes as drilling progressed was logged to help determine the locations of water-bearing zones. The data from these observations, along with pump test data, were used as the basis for determination of the site hydrogeology.

Following the drilling of these three wells, P-1 was pump tested. Analysis of pump test data indicated that pumping had a significant impact on mine inflow, and two additional test wells and two observation wells were located and constructed. Three of the additional wells were over 150 m (500 ft) deep and were drilled to or within 15 m (50 ft) of the "B" coal seam. The fourth, a shallow observation well, was drilled to 58 m (190 ft). All additional wells were drilled and logged in a manner similar to the initial three wells.

Following pump testing of P-1 and prior to any additional testing, all six of the deep wells were partially cased to seal off upper water-bearing strata. A Hookwall packer was attached to the base of the casing: 15 cm (6 in.) diameter casing for test wells and 10 cm (4 in.) diameter casing for the observation wells. The packers were firmly seated in a massive sandstone unit in an attempt to prevent falling water in the wells. The shallow observation well (OB-3) was left uncased to detect drainage effects on shallow rock units. Well construction details are shown on drill logs at the end of Section 7, Figures 16-22 (pages 51-57).

Pump Testing Dewatering Wells - Pump testing was accomplished using a 3.7 kW (5 hp) submersible pump with a gasoline generator as an electrical power source. Constant discharge tests were run for variable lengths of time and water levels were monitored continuously throughout the pumping and recovery periods. Stevens continuous float-operated water-level recorders were installed on the observation wells to monitor changes in water level. Measurement was made through small diameter PVC tubing which was affixed to the pump discharge line and which extended to within 3 m (10 ft) of the pump. Pumpage rates in the pumping well were determined by measuring the length of time necessary to fill a 20 ℓ (5 gal) bucket. Pumping was on the order of 1.5 to 1.9 ℓ/s (25 to 30 gpm), so this was easily accomplished.

P-1 was initially pump tested in November 1976. OB-1 and OB-2 were used as observation wells in this test. All three of these wells were uncased at that time. Pumping in P-1 was at a rate of 1.9 ℓ/s (30 gpm) and continuous for 19 hours. Drawdown measurements were recorded at both the observation wells. Because of problems with the water-level indicator in the pumping well, no drawdown measurements were recorded in P-1. P-1 was again pump tested in

May 1977. All six of the surrounding wells were completed at that time and were used as observation wells. Partial casing was installed in P-1 and all the observation wells except OB-3. Testing was for 2.5 hours at a rate of 1.9 ℓ/s (30 gpm). Drawdown and recovery measurements were recorded in the pumping well and in all of the observation wells.

P-2 was pump tested in January 1977. OB-1, OB-2, OB-3 and P-1 were used as observation wells. Pumping initially was at 1.9 ℓ/s (30 gpm); however, the well went dry in less than 30 minutes. The pumping rate was then reduced to 0.5 ℓ/s (7.9 gpm); the well again went dry. The pumping rate was then reduced to 0.15 ℓ/s (2.3 gpm). Pumping was continuous for one hour. Drawdown and recovery measurements were recorded in the pumping well; however, no effect was seen in any of the surrounding observation wells.

P-3 was pump tested in June 1977. All six of the surrounding wells were used as observation wells. All the wells were partially cased except OB-3. Pumping was at 1.46 ℓ/s (23.1 gpm) and continuous for 8.5 hours. Drawdown and recovery measurements were recorded in both the pumping well and the observation wells.

Selection of Pumps and Generator - On the basis of pump testing and mine inflow data, pumps were selected for the pilot well dewatering scheme. The 3.7 kW (5 hp) pump used previously in testing was installed in Well P-2, which produced minimal pumpage during the testing. A 10 cm (4 in.) diameter 5.6 kW (7.5 hp) pump was installed in Well P-1 and, because a second 10 cm (4 in.) was not available, a 15 cm (6 in.) diameter 5.6 kW (7.5 hp) pump was installed in Well P-3. Installation of the latter pump required the removal of casing from P-3. No significant change in water level occurred following the removal of casing. A diesel Caterpillar generator was used as an electrical source for these pumps.

Water Quality Sampling and Analytical Test Procedures

Samples for water quality analysis were collected at the Main C study area and on the surface periodically throughout the performance of this study. Samples were initially collected to obtain background quality data on mine inflow and on the creek adjacent to the proposed dewatering wells. Later, samples were collected to monitor the effects of pump testing and pilot dewatering. Water discharged from the dewatering wells was also sampled to insure that well discharges met the requirements of the Commonwealth of Pennsylvania and to detect any changes in water quality.

In addition to sampling at the underground study area, mine drainage was periodically sampled from one of the two mine drainage exit sumps (F-14) prior to its entrance into the treatment facility. A sample of mine drainage was also collected for the other exit sump (F-1) and at several of the other underground sumps in an effort to trace water quality in the mine. This latter sampling was not thought to be required for the pilot dewatering study. However, it was relevant to the consideration of alternative methods for water quality control.

A field laboratory was set up near the mine site to provide the capability of analyzing samples shortly after their collection. The field laboratory also provided the ability to obtain certain water quality results quickly throughout the study program. Supplies and equipment in the field facility were provided for performing the following analyses: pH, alkalinity, acidity, sulfate, ferrous iron, and specific conductance.

All analyses were performed using methodology approved by the U.S. Environmental Protection Agency (EPA), Quality Assurance Branch. Alkalinity was analyzed by titration with standard sulfuric acid to an electrometric end point of pH 4.5. Acidity was determined by titration with standard sodium hydroxide to an electrometric end point of pH 8.2. The turbide-metric technique using barium chloride was utilized for sulfate determinations. Ferrous iron was analyzed by using the phenanthroline method in conjunction with a spectrophotometer set at 510 nm and providing a light path of 2 cm. Other analyses, such as total dissolved solids, total iron, calcium, and magnesium were performed at ENCOTEC's main laboratories in Ann Arbor, Michigan. Samples collected for these analyses were preserved according to EPA procedures and shipped to the laboratory for analysis. All metal analyses were performed using atomic absorption spectrophotometry by the flame method.

Abandonment

Following the completion of all field activities, including the pilot well dewatering, the casing in all the wells was pulled and the wells were cement and concrete grouted to the surface. Abandonment procedures were in accordance with the standards set by the Commonwealth of Pennsylvania, Department of Environmental Resources.

HYDROGEOLOGY

Surface Geology

Unconsolidated materials cover most of the surface of the area and include Quaternary alluvium, colluvium, and sedentary soil. Sedentary soil and colluvium make up the greater percentage of this cover and typically consist of silty sand to silty clay. It forms a soil cover over bedrock units that varies widely in thickness. Quaternary alluvial material consists chiefly of sandy silt and sand. This alluvium is restricted to the flood plain and channel of Laurel Lick Run.

All rocks exposed on the surface are of Pennsylvanian age and belong to the Glenshaw Formation of the Conemaugh Group. Typically they consist of thinly to thickly bedded sandstone and shale. Sandstones are generally quartzose and/or argillaceous, fine to very fine grained, moderately hard to very hard, moderately weathered, and iron oxide stained. Shales are gray to gray brown, moderately hard, and moderately to severely weathered. Local residents stated that a thin coal seam outcrops in the immediate site area; however, it was not found in the field. The outcrops of sandstone and shale are relatively few in number and are generally covered by soil units. For this reason, these rock units (with one exception) were not differentiated in mapping and they were grouped as one unit with the colluvium and sedentary soil.

The only mappable bedrock unit was the Saltsburg Sandstone, a member of the Glenshaw Formation of the Conemaugh Group. Outcrops of this unit are generally small; however, a 6 to 7.5 m (20 to 25 ft) vertical section is exposed along a railroad cut in the central portion of the site. Previous geologic mapping by Campbell and others (1913)[5] mapped this unit as the Buffalo Sandstone. Data cited in their report, along with other published geologic literature on the area, and drilling data included in this report were used to identify this rock unit. Figure 11 shows the areal geology in the immediate vicinity of the study area.

Published geologic maps and the geologic mapping of the area indicate that the strata are generally flat-lying, with a slight dip to the southeast. No faults or other linear features were identified in either the field investigation, or in the review of low-altitude photos of the area.

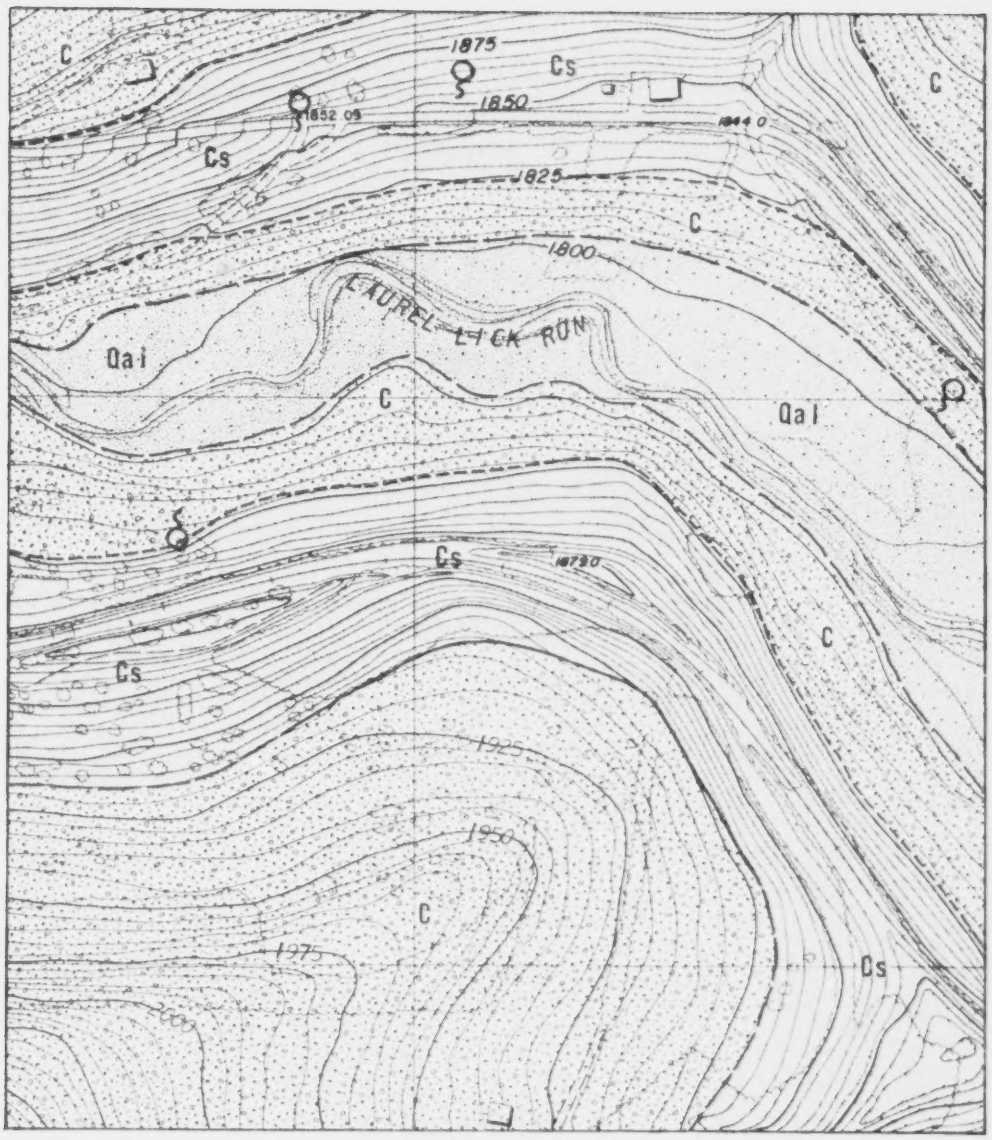
Seeps, springs, and other hydrologic features were also noted during the surface geologic mapping and are identified on the geologic map. Springs were few in number and not associated with any given horizon or topographic elevation. Only flows which were continuous year round were designated as springs, and flows for these were generally greater than 0.3 ℓ/s (5 gpm). Seeps, on the other hand, were quite common. Flow is usually less than 0.05 to 0.1 ℓ/s (1 to 2 gpm), occurs intermittently, and is associated with wet periods. Seeps are commonly found in the topographic lows or swales or in areas with a break in slope.

Subsurface Geology

A stratigraphic column of units encountered in the drilling program is shown on Figure 12. This is a generalized column and represents a composite of all information obtained during the drilling phase.

All rocks encountered in the subsurface are from the Pennsylvanian period and are sedimentary in origin. Lithologically they are sandstone, shale, siltstone, limestone, coal, and clay. Figure 13 is a cross-section of the site based on well data. Correlation between wells was accomplished using several of the coal beds which occur throughout the section. Since drill cuttings rather than cores (for cost reasons) were the source of log data, not every stratigraphic unit was identified in each well, especially some of the thinner beds. Where this was the case, correlation lines have been dashed or queried.

Rocks encountered in the subsurface can be divided into two stratigraphic groups: the Allegheny and the Conemaugh. The Allegheny Group, the older of the two, can be further divided into three or five formations. Flint (1965)[6] subdivided it into three formations (oldest to youngest): the Clarion, the Kittanning, and the Freeport Formations. On the other hand, Edmunds (1969)[7] subdivides it into five formations (oldest to youngest): the Clearfield Creek, the Millstone Run, the Mineral Springs, the Laurel Run, and the Glen Richey. The stratigraphic column (Figure 12) follows Flint's division of the group. The lowermost formation in both Flint's and Edmunds' subdivisions was only partially penetrated--1.5 m (5 ft)--in the borings and is not shown. Maximum thickness encountered in the wells for the Allegheny Group is 76 m (250 ft).



EXPLANATION

GEOLOGIC UNITS

Qal — RECENT ALLUVIUM; FLUVIAL DEPOSITED SAND AND SILT.

C — CONEMAUGH GROUP, MASKED BY COLLUVIAL MATERIAL.

Cs — SALTSBURG SANDSTONE; (MEMBER OF GLENSHAW FORMATION OF CONEMAUGH GROUP) LIGHT YELLOW BROWN TO GRAY BROWN, VERY FINE GRAINED, QUARTZOSE, MODERATELY HARD TO VERY HARD; LOCALLY CROSS-BEDDED; TYPICALLY BREAKS INTO TABULAR FRAGMENTS.

SYMBOLS

SPRING

GEOLOGIC CONTACT; DASHED WHERE APPROXIMATELY LOCATED; SMALLER DASHES WHERE PROJECTED BY DRILL HOLE OR TOPOGRAPHIC DATA.

SCALE

25 0 25 METERS

100 0 100 FEET

CONTOUR INTERVAL 5 FEET

Figure 11. Areal geology.

DESCRIPTION

COLLUVIUM – 0–7.9 m (0–26'); SLOPE WASH DEBRIS; TYPICALLY, MODERATE BROWN TO REDDISH BROWN SILTY–CLAYEY SAND AND CLAYEY SILT.

SALTSBURG SANDSTONE – 0–25.0 m (0–82'); LIGHT BROWN TO GRAY BROWN, VERY FINE TO MEDIUM GRAINED, PEA GRAVEL LOCALLY PRESENT, QUARTOSE, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; WEATHERED NEAR SURFACE, SOME LOCAL MODERATELY WEATHERED, OTHERWISE FRESH, FEW INTERBEDS OF SHALE AND SILTSTONE; EROSIONAL LOWER CONTACT.

SHALE, LIMESTONE – 14.6–18.6 m (48–61'); LIGHT BLUISH GRAY LIMESTONE AND LIMEY SHALE BOUNDED BY UPPER AND LOWER SHALE UNITS. UPPER SHALE: DARK GRAY, MODERATELY HARD. LOWER SHALE: GRAY, WITH SOME RED TO REDDISH BROWN NEAR BASE (MEYERSDALE RED BED?); MODERATELY HARD; INTERBEDS OF SILTSTONE AND SANDSTONE IN BOTH SHALE UNITS.

BUFFALO SANDSTONE–SILTSONE – 2.4–8.8 m (8–29'); GRAY TO GREENISH GRAY, VERY FINE TO FINE GRAINED, MICACEOUS, ARGILLACEOUS, MODERATELY HARD TO HARD.

SHALE – 10.7–15.8 m (35–52'); GRAY TO GRAY BROWN, SANDY, MODERATELY HARD; SOME INTERBEDDED SILTSTONE.

BRUSH CREEK MARINE SHALE – 6.1–8.5 m (20–28'); DARK GRAY, MODERATELY HARD; BLACK ORGANIC MATERIAL COMMON; CONTAINS A THIN LIMESTONE BED 15 cm (6") THICK NEAR BASE.

BRUSH CREEK COAL – 0.3 m (12").

CORINTH SANDSTONE – 5.8–10.4 m (19–34'); GRAY TO LIGHT BROWN, VERY FINE GRAINED, QUARTOSE, VERY HARD; CONTAINS A THIN LIMESTONE BED IN UPPER HALF OF UNIT.

SHALE – 11.3–15.2 m (37–50'); GRAY TO LIGHT BROWN, MODERATELY HARD; INTERBEDS OF SILTSTONE AND VERY FINE GRAINED SANDSTONE; RED SHALE NEAR MIDDLE OF UNIT (MAHONING RED BED?).

MAHONING SANDSTONE – 4.9–7.4 m (16–26'); LIGHT GRAY TO GRAY BROWN, VERY FINE GRAINED, QUARTOSE, VERY HARD, LOCAL INTERBEDS OF GRAY TO GRAY BROWN SHALE, GRADATIONAL CONTACT WITH OVERLYING SHALE.

UPPER FREEPORT LIMESTONE – 4.9–5.8 m (16–19'); DARK GRAY SHALE IN LATERAL CONTACT WITH GRAY LIMESTONE AND LIMEY SHALE OF VARIABLE THICKNESS; EROSIONAL UPPER CONTACT.

INTERBEDDED SHALE, SILTSTONE AND SANDSTONE – 13.7–15.8 m (45–52'); GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MICACEOUS, MODERATELY HARD TO VERY HARD; CONTAINS POSSIBLE THIN COAL SEAM <15 cm (6") THICK 3.0 m (10') ABOVE BASE.

LOWER FREEPORT ("D") COAL – 0.3–1.2 m (12–48").

INTERBEDDED SHALE, SILTSTONE AND SANDSTONE – 11.0–11.9 m (36–39'); GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; POSSIBLY CONTAINS 2 THIN COAL SEAMS <15 cm (6") THICK.

UPPER KITTANNING ("C PRIME") COAL – 0.5–0.9 m (18–36").

UPPER WORTHINGTON SANDSTONE – 11.3–11.6 m (37–38'); GRAY, VERY FINE TO FINE GRAINED, QUARTOSE, VERY HARD; MINOR SHALEY INTERBEDS.

SHALE – 9.8–11.0 m (32–36'); GRAY TO DARK GRAY; SOME SILTSTONE INTERBEDS; POSSIBLE THIN COAL SEAM AT TOP.

MIDDLE KITTANNING ("C") COAL – 0.5–0.7 m (18–26").

LOWER WORTHINGTON SANDSTONE – 17.7 m (58'); GRAY, VERY FINE GRAINED, QUARTOSE, VERY HARD; MINOR GRAY SHALE AT TOP AND NEAR BASE.

LOWER KITTANNING ("B") COAL – 1.5± m (60±").

Figure 12. Site stratigraphic column.

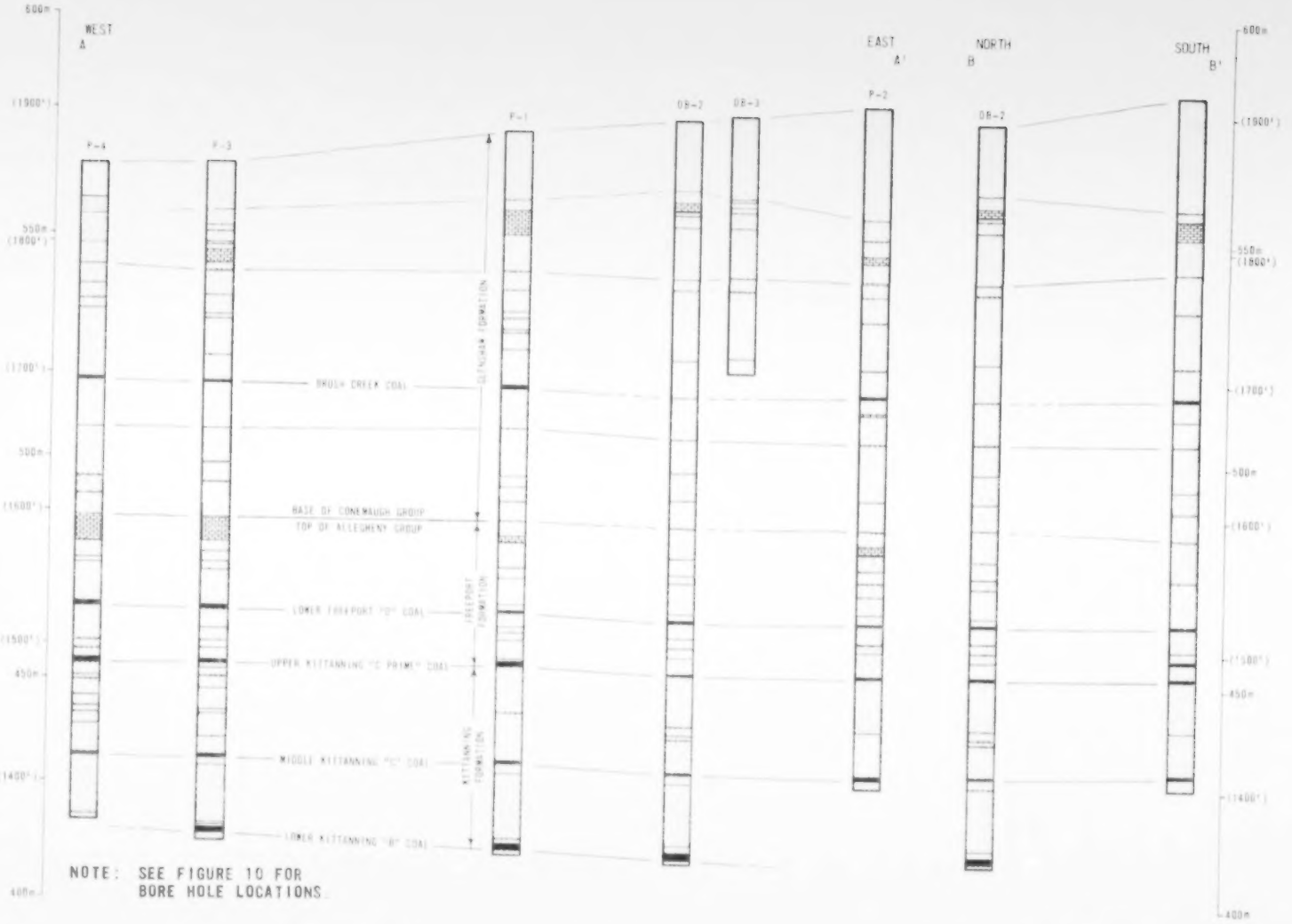


Figure 13. Site cross-section.

The Conemaugh Group overlies the Allegheny and can be divided into two formations: The Glenshaw and the Casselman. The Casselman Formation lies stratigraphically higher than the Glenshaw and was not encountered on the surface nor in the subsurface. Maximum thickness encountered for the Glenshaw Formation is 100 m (328 ft). Names of members for both the Allegheny and the Conemaugh Groups, along with relative thicknesses, are indicated on the stratigraphic column. Names are taken from published literature and are based on distances from known marker beds.

The Allegheny Group is of local and regional economic importance because of the included coal seams. Of the major seams regionally mined, four of the five were encountered in the drilling. From the oldest to youngest, they were: Lower Kittanning or "B" coal, Middle Kittanning or "C" coal, Upper Kittanning or "C prime" coal, and the Lower Freeport or "D" coal. All these coals consist of one or more seams with a variable amount of shale or boney coal separating them. The Upper Freeport or "E" coal was the only regionally important coal not identified or not of sufficient thickness to be recorded in any of the borings. Flint (1965)[8] states that it is locally absent in Somerset County because of an erosional unconformity, and this may be a possible explanation for its absence in the present study area also. Coal is only of local importance in this section of the Conemaugh Group and only one seam was encountered in the borings. This was the Brush Creek coal.

Structurally the area is generally flat-lying with a gentle dip to the east-southeast (see Figure 13). Strata lie subparallel to one another except for the lowermost coal seam, the Lower Kittanning (a member of the Kittanning Formation of the Allegheny Group), which dips to the east at a slightly greater angle. Minor flexuring is present, and no fault features were identified by correlation of geologic units.

Groundwater Conditions

Attempts were made to identify the principal water-bearing zones as they were encountered during air-rotary drilling by logging the amount of water air-lifted as the holes advanced in conjunction with logging of rock types. Each water-bearing zone encountered should generally increase the amount of water air-lifted as long as circulation is maintained. It was found generally there was poor correlation between rock types and the amount of groundwater encountered. There appears to be some perching of groundwater in the Saltsburg Sandstone and possibly in the underlying unidentified limestone. Below this apparent perched zone, the bedded rock units could not be separated into distinct aquifers or zones with any degree of consistency. In particular, the various sandstone beds encountered did not indicate higher permeabilities than even the shales. There were some indications that coal seams and the Upper Freeport Limestone are relatively permeable, but not with consistency. Also, there were indications that the thin shale bed above the Lower Kittanning ("B") coal which forms the mine roof in the Main G study area, is relatively permeable. This information, although not conclusive in itself, led to the suspicion that fractures controlled the groundwater flow regime.

After the first test well and two observation wells were completed, pump tests were run. The well (P-1) was left open hole for the first test; later, a blank casing was installed to a depth of 65.5 m (215 ft) with a packer at the bottom of the casing to seal the upper part of the well, and another pump test was performed. The transmissivities and storativities calculated from observation well data were essentially the same for both tests. Well P-1 was tested at a rate of 1.9 ℓ/s (30 gpm) for 19 hours with a maximum drawdown of 40.5 m (133 ft). The drawdown response in the two observation wells was erratic with respect to time and distance from the pumped well. Transmissivities calculated from log-log time-drawdown plots from observation well data were inconsistent with time-drawdown and recovery data from the pumped well. Observation well data indicated transmissivities more than an order of magnitude higher than data from the pumped well. Distance drawdown plots provided closer agreement with pumped well data, but only when each observation well was plotted separately with respect to the drawdown in the pumped well. This was because drawdown response was very asymmetric and the cone of depression around the pumped well was elongated along fractures, possibly in several directions.

This pump test data, along with drilling information and observation of inflows to the study area underground, all reinforced the opinion that flow paths were controlled by fracture zones. Also, it was observed that there was very rapid response to heavy rainfall both in water levels in wells and in inflows to the mine, indicating hydraulic connection of fracture zones with the surface. This presented a problem in locating and spacing dewatering wells because zones of fracturing were not recognizable. The only evidence for concentration of fracturing was the localization of water inflows to the mine in the Main G study area. On the surface, the prominent joint sets strike to the northwest and northeast. Therefore, it was decided to construct the remaining wells to the north of the underground study area along an east-west alignment as shown on Figure 10. Since pump tests of P-1 indicated a substantial effect on the monitored inflow, the new wells were placed at approximately the same distance--30 m (100 ft)--from the northern mine face.

Drilling of these wells indicated variable yields and it was decided to pump test each well individually. Well P-2 yielded only about 0.06 ℓ/s (1 gpm) at the maximum available drawdown and was abandoned for use as a dewatering well. Well P-3 was pumped at 1.46 ℓ/s (23.1 gpm) for 8.5 hours with a drawdown of 14.2 m (46.7 ft). Well P-4 was not tested because originally it was designed as an observation well. However, drilling information indicated a good potential yield, and it was used as a dewatering well during the pilot dewatering operation.

Using time-drawdown data from the pumped wells (P-1 and P-3) and distance drawdown data, transmissivities were calculated that ranged between 2.5 and 3.7 $m^3/m \cdot d$ (200 to 300 gpd/ft). This was later confirmed using data developed from the pilot dewatering operation and applying a graphical method developed by Theis and Conover (1963)[9] to determine the percentage of pumped well water being diverted from a drain. Since the percentage of pumped water being diverted from the drain (mine inflow) was known, along with the period of pumping and distance from the drain, the value of S/T was determined graphically (where T = transmissivity and S = storativity). Since the groundwater appears

to be unconfined, a storage constant of 0.03 to 0.06 was estimated (storage constants could not be calculated from pump tests). Solving for transmissivity yielded approximate values ranging from 1.2 to 3.7 $m^3/m \cdot d$ (100 to 300 gpd/ft), which agrees well with pump test data.

Water level contours were constructed with the limited well data as shown on Figure 14. These contours are estimates because of poor control along the mine face and north of the line of wells, but they illustrate the probable flow pattern into the mine study area prior to the pilot dewatering operation in September 1977. As constructed, they indicate two fracture zones draining into the mine study area. These fracture zones strike in about the same direction as the predominant joint sets in the area and they appear to intersect in the area of the highest inflow to the mine. The wider of the two zones may be the one striking to the northwest. It appears that all of the pumping wells missed the most permeable parts of these zones. In retrospect, it appears that OB-2 should have been used as a dewatering well and P-3 should have been located about 18 m (60 ft) to the east. This illustrates the need for a practical method and approach for detecting fracture zones to guide drilling operations.

Hydraulic interconnection between fracture zones and even within a fracture zone across fracture planes is variable at best. For example, when P-1 was pump tested after all the wells were constructed, much larger responses to pumping (drawdowns) were recorded in wells P-3 and P-4 than in OB-1 and OB-2, which were much closer. When P-3 was pump tested, a much larger response was recorded in OB-2 than in either P-1 or P-4, which were both much closer. Also, when P-2 was pump tested, there was no response in any of the other wells. From this, one could speculate in retrospect that during the dewatering operation very little of the water flowing into the mine along the northeast-trending fracture zone was intercepted. In fact, flows may have increased due to steeper gradients induced by reductions in flow along the northwest fracture zone.

In conclusion, data strongly indicate that groundwater drains into the Main G study area along narrow preferred flow paths. These flow paths are controlled by fractures and are directionally oriented. Permeabilities across these flow paths may be orders of magnitude lower than permeabilities along the fracture planes. The bedded rock units probably act as secondary aquifers to various degrees and provide some hydraulic interconnection between fractures, as well as providing groundwater storage space. The fracture zones appear to consist mainly of steeply dipping fractures that intersect the surface. This permits rapid recharge from direct precipitation and from streams or ponds at the surface. The secondary aquifers also release ground water from storage to the fracture zones. The groundwater flow regime is therefore quite complex and difficult to model with mathematical techniques because of the severe boundary effects and directional variations in permeability. In this fracture-dominated flow system, groundwater response to dewatering wells is nearly impossible to predict with confidence unless the fracture zones are defined and their hydraulic properties determined. The amount of exploration work required to do this may be impractical unless the fracture zones can at least be defined by methods such as aerial photography combined with surface

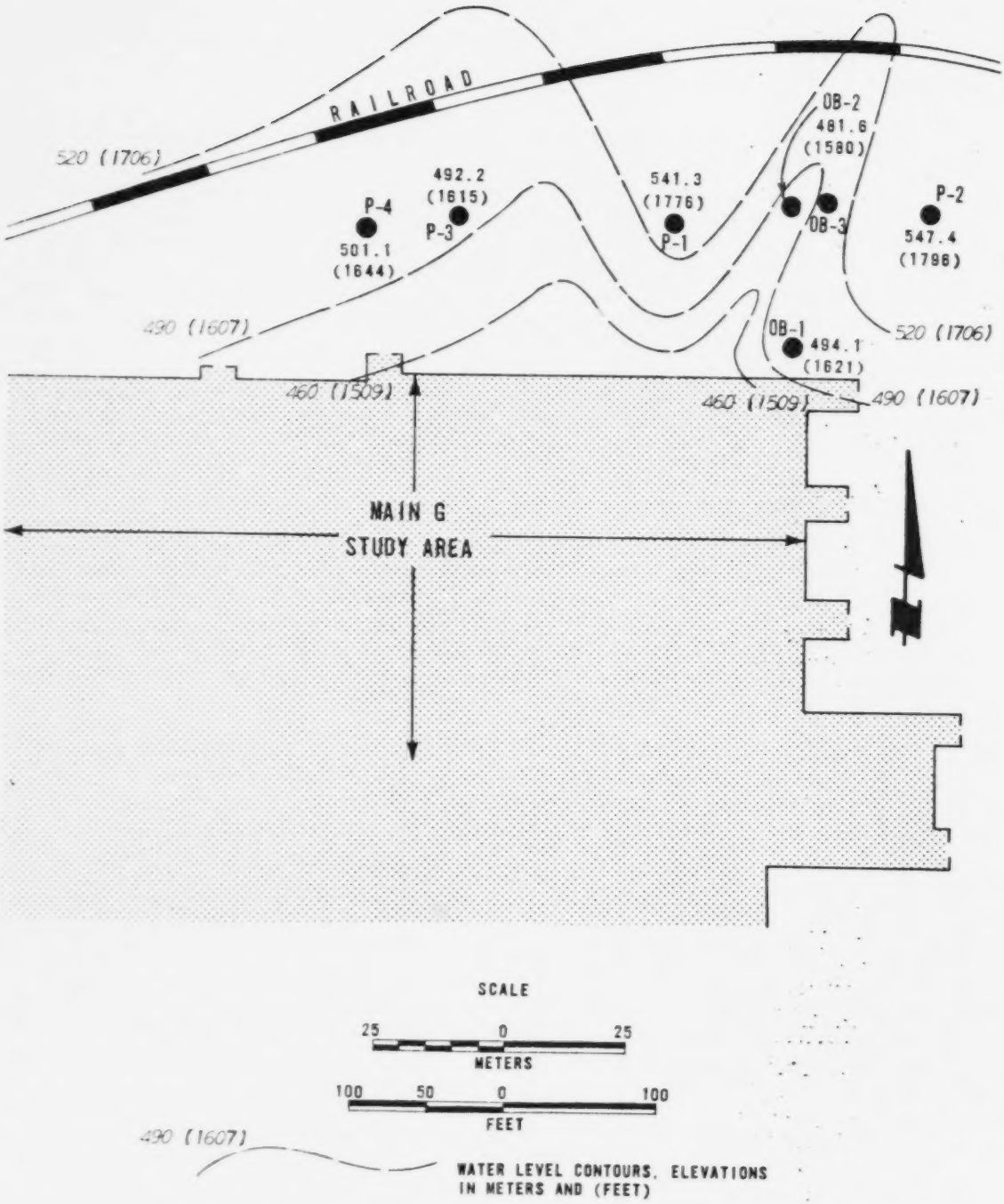


Figure 14. Well locations and water level contours 9/15/77.

geophysical techniques such as resistivity or electric self potential surveys to supplement test wells. More field-oriented research is needed to determine the practical application of these and other techniques to mine dewatering.

BASELINE DATA

Baseline data was collected to: (1) establish water flow data into and out of the Main G study area and the mine; and (2) develop background water quality data in the mine, from test wells, and from Laurel Lick Run. In general, water inflows to the mine and background water qualities were about as expected when the site was selected. Inflows to the Main G study area were at the lower end of the range originally estimated--6.3 to 9.5 ℓ/s (100 to 150 gpm). Degradation in water quality was found to be slower than anticipated as the water was transported through the mine. As will be discussed further in later sections, it was found that the most seriously degraded water comes from abandoned or inactive portions of the mines. This was not investigated directly but it can easily be deduced from the data developed. Background water qualities for groundwater from the study area and from Laurel Lick Run were about as expected.

Underground Flow

Flow measurements at the underground monitoring station began in the late part of September, 1976. Figure 15 represents typical background flow data for the total water inflow occurring in the Main G study area of the mine. The data shown are average weekly values and the standard deviation for each 7-day period. These data are based on average daily flows derived from individual flow meter readings, and have been adjusted for changes in L-2 sump water levels where appropriate.

The average inflow rate for the entire period of September 26 through June 5, 1976 was 6.69 ℓ/s (106 gpm) with a standard deviation of $\pm$0.57 ℓ/s (9 gpm). As can be seen on Figure 15, the water inflow did vary somewhat over time, but such changes were generally gradual in nature. Some fluctuation in measured flow did occur from day to day but usually followed a given increasing or decreasing trend over several days. Changes occurred in conjunction with wet (heavy rain or snowmelt) periods and dry periods. It did not appear that major fluctuations in flow occurred within a given 24-hour period. Variations in the data on a very short term basis (less than one day) were more a function of variations within the pump transferring the water through the meter than of actual large changes in water inflows into the mine. Background flow monitoring in the Main G study area continued until the first pilot dewatering period began in July, 1977. The data developed are similar to those presented on Figure 15.

The total mine discharge to the treatment plant, averaging 13 000 m^3/d (3.4 mgd), was discussed previously, and the average monthly flows are shown on Figure 15. The primary source of data regarding the mine water treatment plant flow was the pumping logs of the pumps in the F-1 and F-14 sumps. These logs contained information on the hours each pump ran, and flow to the treatment plant was then determined based on the hours each pump ran and on the respective pump curves for each pump. Records of all pumps connected to the

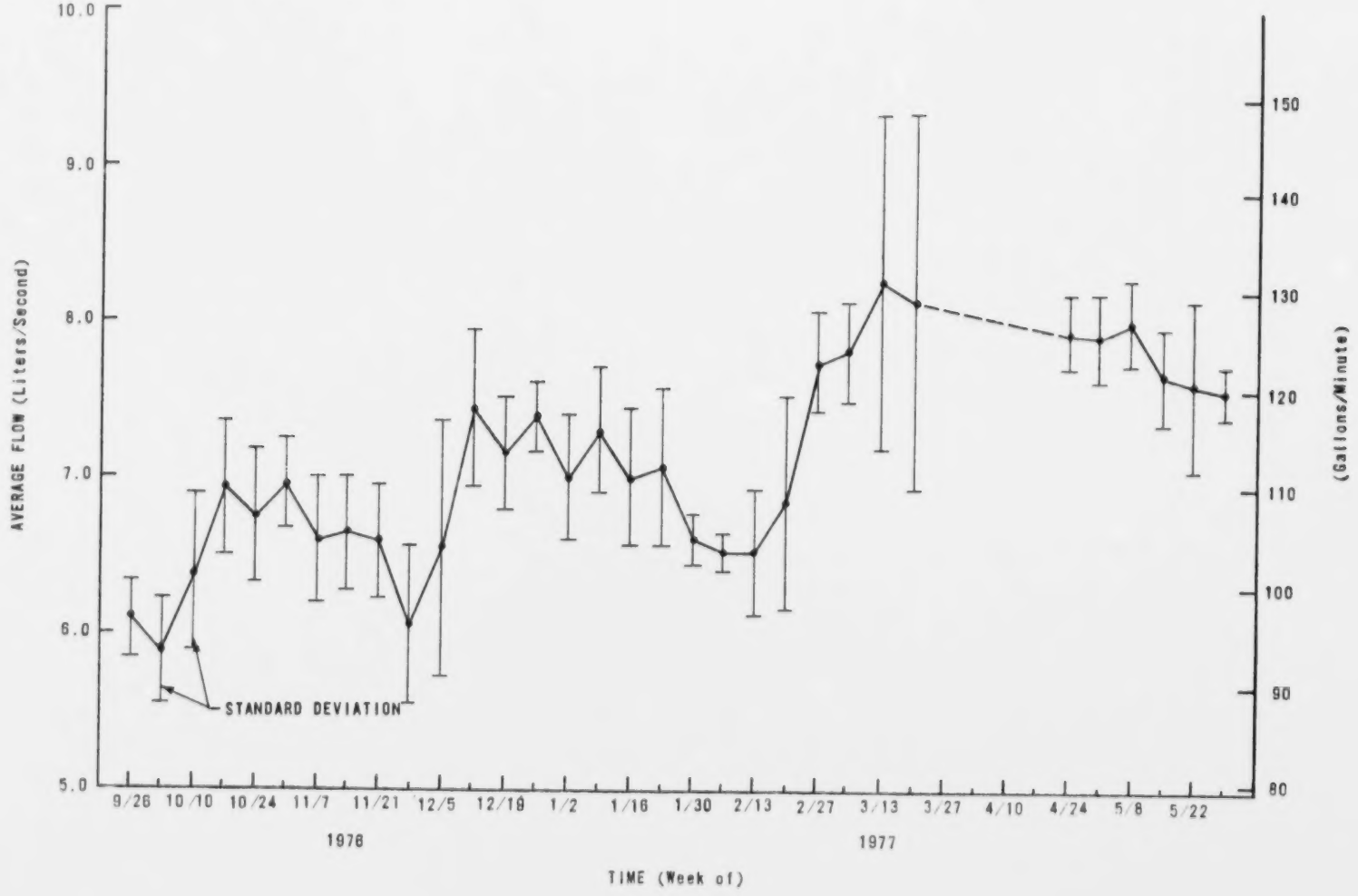


Figure 15. Average mine inflow - Main G study area 9/26/76 to 6/5/77.

treatment plant were available and checked periodically by the study team.

The F-14 line carrying water to the treatment plant had one location near the surface where a water meter could be installed. The meter was installed in this line from November 1976 until April 1977. These data were obtained in order to augment and check the pump curve estimates. A similar meter could not be installed on the F-1 line due to its location. Hence, the accuracy of using pumping logs and pump curve characteristics to predict flow rates could not be evaluated for this line.

Water Quality

Water samples were collected and analyzed to provide baseline water quality data in the Main G study area and at selected points through the mine before the mine water is treated and discharged. The results of this sampling are presented in Table 4, Baseline Water Quality Data. Samples of both roof inflow and L-2 sump water were collected to compare quality of the ground water entering the mine with the quality of the water at the primary underground monitoring station. In establishing the L-2 monitoring station, it was assumed that water quality would not deteriorate significantly in the Main G study area before it is transferred to other parts of the mine. Comparison of the results of this sampling for these two locations shows that the waters were generally similar and that the assumption was valid. Roof samples were discontinued after November 1976. The L-2 sampling station utilized the Protech sampler (see Figure 8) and mine personnel collected samples except when this device was periodically serviced.

A series of samples was collected at sumps to trace the quality of mine water after it leaves the Main G study area to the treatment plant on the surface. Single samples were obtained in October 1976 from the G-7 sump, slant sump, F-1 and F-14 discharge lines. Additional samples were collected from the F-14 discharge line later in the study. For safety reasons, our team members did not have access to the F-1 line sampling point and the single sample was obtained by mine personnel. Results of the analysis of the F-1 sample are as follows:

pH	3.03
Acidity (mg/ℓ as $CaCO_3$)	105
Sulfate (mg/ℓ)	880
Total Iron (mg/ℓ)	50

The analyses of the samples from the G-7 and the slant sump illustrate the deterioration in quality as the mine water is transported from area to area and finally discharged to the treatment plant. Water quality is still good at the G-7 sump with respect to iron, sulfates and pH. At the slant sump, after mixing with mine water from other areas including abandoned workings, the quality is poor and would require treatment before it could be discharged to a receiving stream. Water quality deteriorates further while passing through the F-14 and F-1 sumps and before reaching the treatment plant.

The water samples from Laurel Lick Run were collected at Station A, located upstream from the pilot dewatering area. The alkalinity levels were generally very low in this creek, indicating a weakly buffered system. On most occasions the pH remained above 6.5, but on one day (October 6, 1976) the pH dropped to 3.2. The variation in other parameters was also considerable and may reflect the influence of coal waste deposits upstream, along with variations in flow.

One sample of groundwater was obtained from Well P-1 during the initial pump testing. This sample was analyzed to confirm that the quality of water pumped from the dewatering wells would be similar to the roof water in the Main G study area and that direct discharge to Laurel Lick Run would be acceptable. Analysis of this sample (see Table 4) confirms that the groundwater quality is similar to mine water at the inflow point and suitable for direct discharge to Laurel Lick Run.

TABLE 4. BASELINE WATER QUALITY DATA

Location	Date 1976	Sample Type	pH	Alkalinity mg/ℓ $CaCO_3$	Acidity mg/ℓ $CaCO_3$	Specific Conductance µS/cm @25°C	Sulfate mg/ℓ	Total Dissolved Solids mg/ℓ	Calcium mg/ℓ	Magnesium mg/ℓ	Ferrous Iron mg/ℓ	Total Iron mg/ℓ	Total Suspended Solids mg/ℓ	Potassium mg/ℓ	Sodium mg/ℓ	Chloride	Sulfide
Main G Study Area-Roof Water	5/21	Grab	8.04	115	--	340	41	--	--	--	--	1.59	--	--	--	--	--
	9/23	Grab	8.49	134	--	240	--	190	--	--	--	0.01	--	--	--	--	--
	10/5	Grab	8.1	107	--	270	43	230	23	6.2	0.05	0.08	2	1.8	36	4	2.1
	10/7	Grab	8.1	118	--	230	21	220	6	1.7	<0.02	<0.01	1	0.8	63	5	1.6
	11/22	Grab	8.4	118	--	235	19	--	--	--	<0.02	--	--	--	--	--	--
	11/24	Grab	8.2	116	--	--	--	--	--	--	--	--	--	--	--	--	--
Underground L-2 Station	9/23	Grab	8.03	127	--	272	--	210	--	--	--	0.87	--	--	--	--	--
	10/5	Grab	8.0	111	--	270	33	230	16	4.0	0.03	0.01	2	1.3	51	4	<0.01
	10/6	Comp.	8.0	114	--	260	79	220	17	4.1	0.14	0.17	9	1.3	51	--	--
	10/7	Comp.	8.0	117	--	250	67	230	16	4.2	0.11	0.09	15	1.4	53	5	<0.01[a]
	11/22	Grab	7.9	108	--	248	29	--	--	--	<0.02	--	--	--	--	--	--
	11/23	Comp.	8.2	112	--	--	43	--	--	--	--	--	--	--	--	--	--
	11/24	Comp.	8.11	113	--	--	--	--	--	--	--	--	--	--	--	--	--
	11/24	Grab	7.9	112	--	--	--	--	--	--	--	--	--	--	--	--	--
Laurel Lick Run-Station A	6/-	Grab	7.4	26	--	160	44	--	--	--	--	1.4	--	--	--	--	--
	10/5	Grab	7.0	12	--	130	33	130	12	4.6	0.04	0.35	3	2.0	6.2	7	<0.01
	10/6	Grab	3.2	--	31	300	74	140	13	4.6	<0.02	0.35	3	1.8	6.1	--	<0.01
	10/7	Grab	6.3	9	--	100	71	180	12	4.5	0.03	0.26	6	1.8	5.6	6	--
	11/22	Grab	7.0	17	--	380	190	--	--	--	0.18	--	--	--	--	--	--
	11/23	Grab	7.2	18	--	--	185	--	--	--	--	--	--	--	--	--	--
Treatment Plant Influent Line F-14	7/-	Grab	6.27	15	--	--	610	--	--	--	--	12	--	--	--	--	--
	10/7	Grab	4.4	--	309	1850	240	2310	152	54	230	134	--	--	--	--	--
	11/22	Grab	6.7	58	--	1150	530	--	--	--	18	--	--	--	--	--	--
	11/23	Grab	6.4	49	--	--	560	--	--	--	--	--	--	--	--	--	--
G-7 Sump	10/7	Grab	7.8	97	--	450	190	450	26	6.8	0.03	0.30	39	1.6	92	--	--
Slant Sump	10/7	Grab	6.7	30	--	1550	210	1720	105	39	14	33	52	4.0	490	--	--
Well P-1	11/30	Grab	8.30	117	--	264	19	156	15	3.4		0.007	2	1.2	46	4	--

[a]This parameter from grab sample.

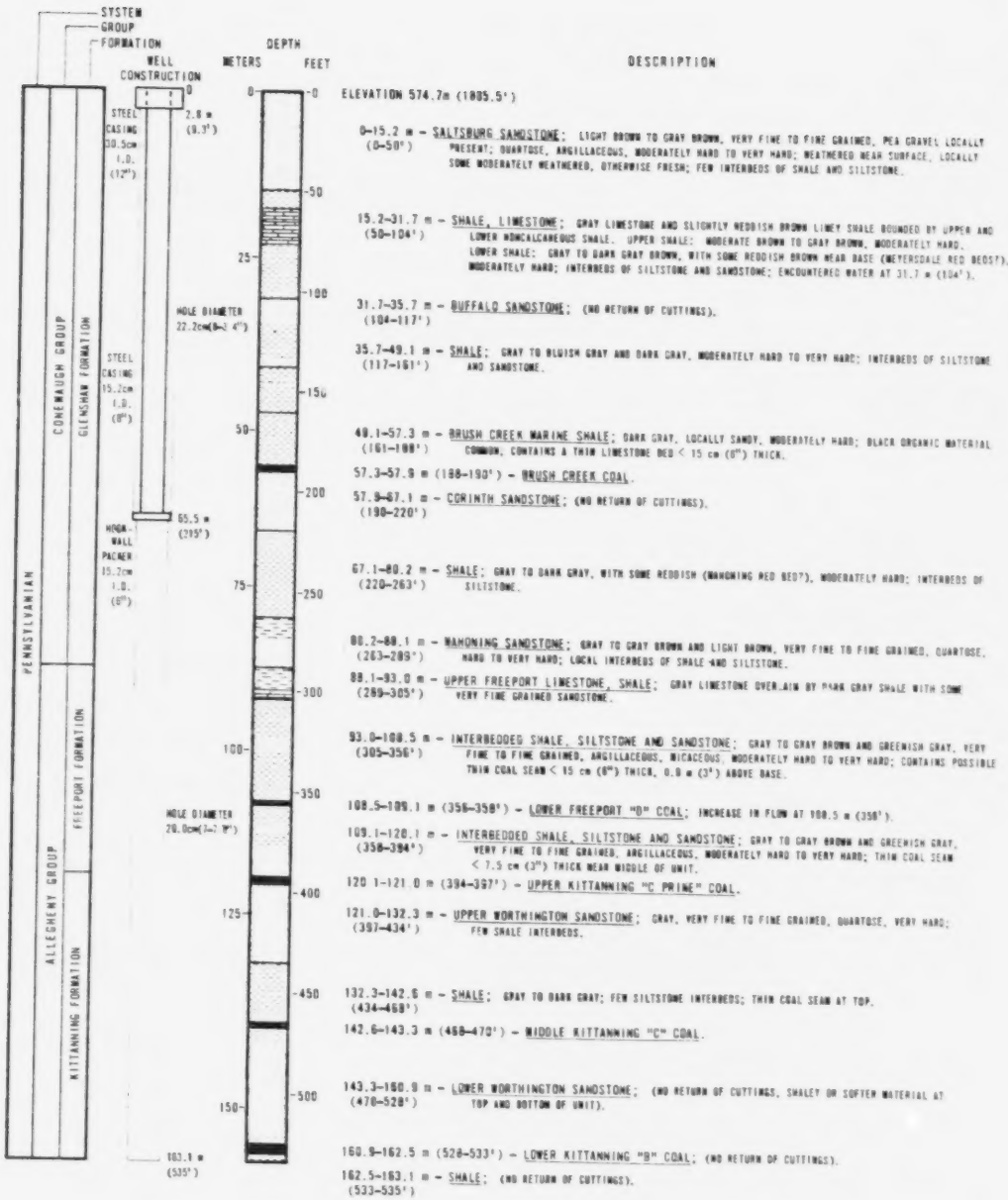

SYSTEM
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
PENNSYLVANIAN
ALLEGHENY GROUP
CONEMAUGH GROUP
KITTANNING FORMATION
FREEPORT FORMATION
GLENSHAW FORMATION
STEEL CASING 30.5cm I.D. (12")
2.8 m (9.3')
HOLE DIAMETER 22.2cm(8-3/4")
STEEL CASING 15.2cm I.D. (8")
65.5 m (215')
HYDRA-WALL PACKER 15.2cm I.D. (6")
HOLE DIAMETER 20.0cm(7-7/8")
163.1 m (535')
ELEVATION 574.7m (1885.5')
0-15.2 m (0-50') - SALTSBURG SANDSTONE; LIGHT BROWN TO GRAY BROWN, VERY FINE TO FINE GRAINED, PEA GRAVEL LOCALLY PRESENT; QUARTOSE, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; WEATHERED NEAR SURFACE, LOCALLY SOME MODERATELY WEATHERED, OTHERWISE FRESH; FEW INTERBEDS OF SHALE AND SILTSTONE.
15.2-31.7 m (50-104') - SHALE, LIMESTONE; GRAY LIMESTONE AND SLIGHTLY REDDISH BROWN LIMEY SHALE BOUNDED BY UPPER AND LOWER NONCALCAREOUS SHALE. UPPER SHALE: MODERATE BROWN TO GRAY BROWN, MODERATELY HARD; LOWER SHALE: GRAY TO DARK GRAY BROWN, WITH SOME REDDISH BROWN NEAR BASE (MEYERSDALE RED BED?), MODERATELY HARD; INTERBEDS OF SILTSTONE AND SANDSTONE; ENCOUNTERED WATER AT 31.7 m (104').
31.7-35.7 m (104-117') - BUFFALO SANDSTONE; (NO RETURN OF CUTTINGS).
35.7-49.1 m (117-161') - SHALE; GRAY TO BLUISH GRAY AND DARK GRAY, MODERATELY HARD TO VERY HARD; INTERBEDS OF SILTSTONE AND SANDSTONE.
49.1-57.3 m (161-188') - BRUSH CREEK MARINE SHALE; DARK GRAY, LOCALLY SANDY, MODERATELY HARD; BLACK ORGANIC MATERIAL COMMON, CONTAINS A THIN LIMESTONE BED < 15 cm (6") THICK.
57.3-57.9 m (188-190') - BRUSH CREEK COAL.
57.9-67.1 m (190-220') - CORINTH SANDSTONE; (NO RETURN OF CUTTINGS).
67.1-80.2 m (220-263') - SHALE; GRAY TO DARK GRAY, WITH SOME REDDISH (MAHONING RED BED?), MODERATELY HARD; INTERBEDS OF SILTSTONE.
80.2-88.1 m (263-289') - MAHONING SANDSTONE; GRAY TO GRAY BROWN AND LIGHT BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, HARD TO VERY HARD; LOCAL INTERBEDS OF SHALE AND SILTSTONE.
88.1-93.0 m (289-305') - UPPER FREEPORT LIMESTONE, SHALE; GRAY LIMESTONE OVERLAIN BY DARK GRAY SHALE WITH SOME VERY FINE GRAINED SANDSTONE.
93.0-108.5 m (305-356') - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MICACEOUS, MODERATELY HARD TO VERY HARD; CONTAINS POSSIBLE THIN COAL SEAM < 15 cm (6") THICK, 0.9 m (3') ABOVE BASE.
108.5-109.1 m (356-358') - LOWER FREEPORT "D" COAL; INCREASE IN FLOW AT 108.5 m (356').
109.1-120.1 m (358-394') - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; THIN COAL SEAM < 7.5 cm (3") THICK NEAR MIDDLE OF UNIT.
120.1-121.0 m (394-397') - UPPER KITTANNING "C PRIME" COAL.
121.0-132.3 m (397-434') - UPPER WORTHINGTON SANDSTONE; GRAY, VERY FINE TO FINE GRAINED, QUARTOSE, VERY HARD; FEW SHALE INTERBEDS.
132.3-142.6 m (434-468') - SHALE; GRAY TO DARK GRAY; FEW SILTSTONE INTERBEDS; THIN COAL SEAM AT TOP.
142.6-143.3 m (468-470') - MIDDLE KITTANNING "C" COAL.
143.3-160.9 m (470-528') - LOWER WORTHINGTON SANDSTONE; (NO RETURN OF CUTTINGS, SHALEY OR SOFTER MATERIAL AT TOP AND BOTTOM OF UNIT).
160.9-162.5 m (528-533') - LOWER KITTANNING "B" COAL; (NO RETURN OF CUTTINGS).
162.5-163.1 m (533-535') - SHALE; (NO RETURN OF CUTTINGS).

Figure 16. Drilling log: Dewatering well P-1.

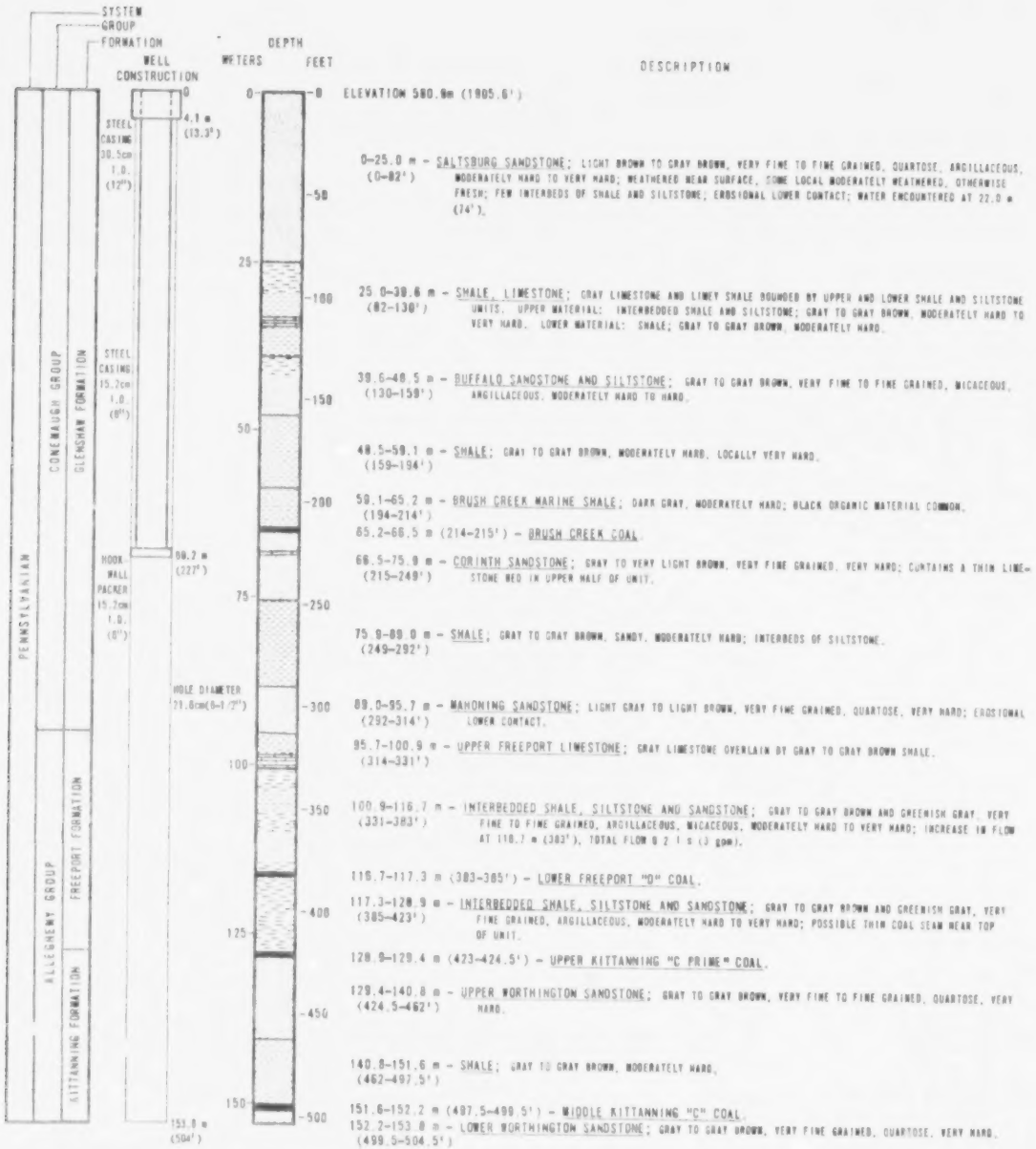

SYSTEM
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
PENNSYLVANIAN
CONEMAUGH GROUP
GLENSHAW FORMATION
ALLEGHENY GROUP
FREEPORT FORMATION
KITTANNING FORMATION
STEEL CASING 30.5cm I.D. (12")
4.1 m (13.3')
STEEL CASING 15.2cm I.D. (8")
HOOK WALL PACKER 15.2cm I.D. (8")
69.2 m (227')
HOLE DIAMETER 21.0cm(8-1/2")
153.0 m (504')
ELEVATION 580.9m (1905.6')
0-25.0 m - SALTSBURG SANDSTONE; LIGHT BROWN TO GRAY BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; WEATHERED NEAR SURFACE, SOME LOCAL MODERATELY WEATHERED, OTHERWISE FRESH; FEW INTERBEDS OF SHALE AND SILTSTONE; EROSIONAL LOWER CONTACT; WATER ENCOUNTERED AT 22.0 m (74').
(0-82')
25.0-39.6 m - SHALE, LIMESTONE; GRAY LIMESTONE AND LIMEY SHALE BOUNDED BY UPPER AND LOWER SHALE AND SILTSTONE UNITS. UPPER MATERIAL: INTERBEDDED SHALE AND SILTSTONE; GRAY TO GRAY BROWN, MODERATELY HARD TO VERY HARD. LOWER MATERIAL: SHALE; GRAY TO GRAY BROWN, MODERATELY HARD.
(82-130')
39.6-48.5 m - BUFFALO SANDSTONE AND SILTSTONE; GRAY TO GRAY BROWN, VERY FINE TO FINE GRAINED, MICACEOUS, ARGILLACEOUS, MODERATELY HARD TO HARD.
(130-159')
48.5-59.1 m - SHALE; GRAY TO GRAY BROWN, MODERATELY HARD, LOCALLY VERY HARD.
(159-194')
59.1-65.2 m - BRUSH CREEK MARINE SHALE; DARK GRAY, MODERATELY HARD; BLACK ORGANIC MATERIAL COMMON.
(194-214')
65.2-66.5 m (214-215') - BRUSH CREEK COAL.
66.5-75.9 m - CORINTH SANDSTONE; GRAY TO VERY LIGHT BROWN, VERY FINE GRAINED, VERY HARD; CONTAINS A THIN LIMESTONE BED IN UPPER HALF OF UNIT.
(215-249')
75.9-89.0 m - SHALE; GRAY TO GRAY BROWN, SANDY, MODERATELY HARD; INTERBEDS OF SILTSTONE.
(249-292')
89.0-95.7 m - MAHONING SANDSTONE; LIGHT GRAY TO LIGHT BROWN, VERY FINE GRAINED, QUARTOSE, VERY HARD; EROSIONAL LOWER CONTACT.
(292-314')
95.7-100.9 m - UPPER FREEPORT LIMESTONE; GRAY LIMESTONE OVERLAIN BY GRAY TO GRAY BROWN SHALE.
(314-331')
100.9-116.7 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MICACEOUS, MODERATELY HARD TO VERY HARD; INCREASE IN FLOW AT 110.7 m (383'), TOTAL FLOW 0.2 l/s (3 gpm).
(331-383')
116.7-117.3 m (383-385') - LOWER FREEPORT "D" COAL.
117.3-128.9 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; POSSIBLE THIN COAL SEAM NEAR TOP OF UNIT.
(385-423')
128.9-129.4 m (423-424.5') - UPPER KITTANNING "C PRIME" COAL.
129.4-140.8 m - UPPER WORTHINGTON SANDSTONE; GRAY TO GRAY BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, VERY HARD.
(424.5-462')
140.8-151.6 m - SHALE; GRAY TO GRAY BROWN, MODERATELY HARD.
(462-497.5')
151.6-152.2 m (497.5-499.5') - MIDDLE KITTANNING "C" COAL.
152.2-153.0 m - LOWER WORTHINGTON SANDSTONE; GRAY TO GRAY BROWN, VERY FINE GRAINED, QUARTOSE, VERY HARD.
(499.5-504.5')

Figure 17. Drilling log: Dewatering well P-2.

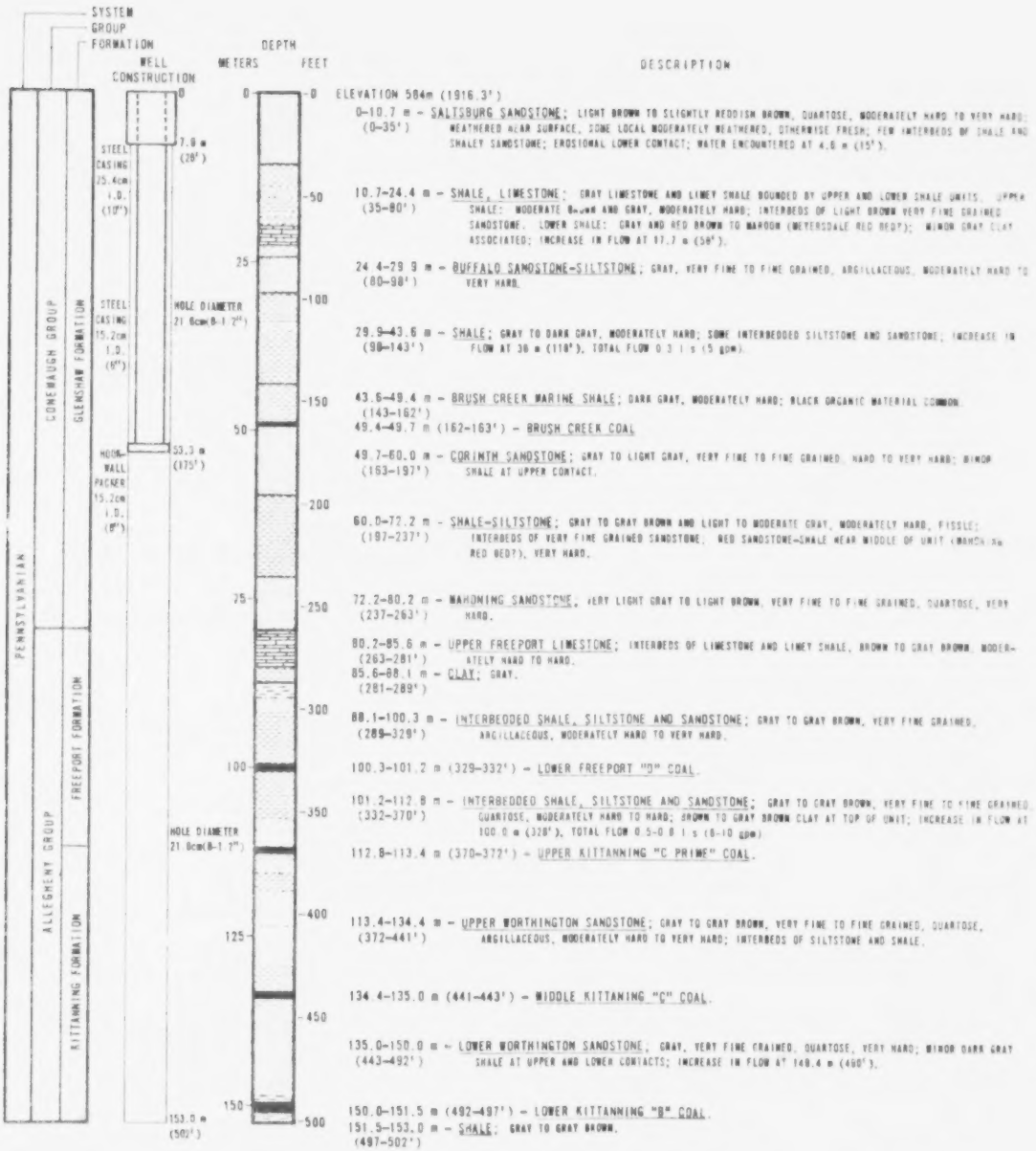
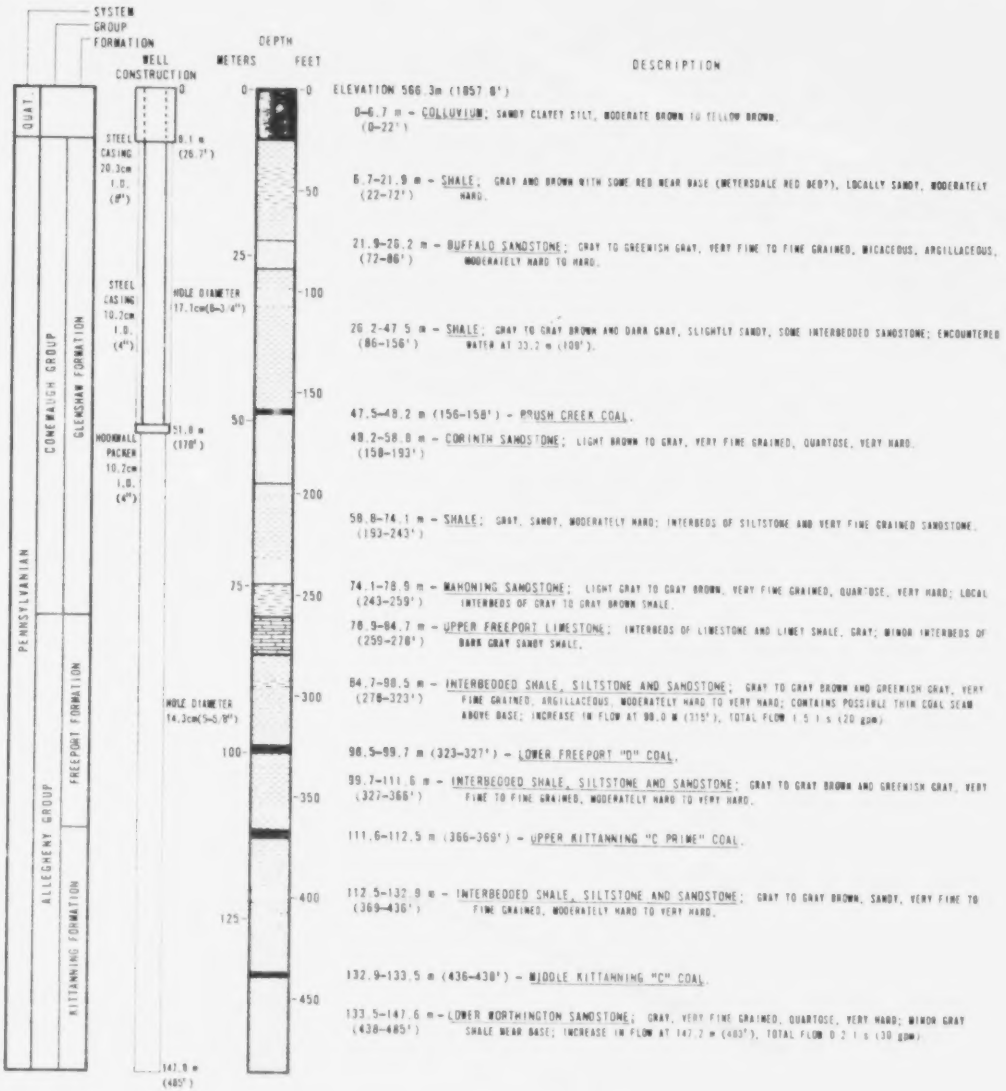

SYSTEM
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
PENNSYLVANIAN
ALLEGHENY GROUP
CONEMAUGH GROUP
KITTANNING FORMATION
FREEPORT FORMATION
GLENSHAW FORMATION
STEEL CASING 25.4cm I.D. (10")
7.8 m (26')
STEEL CASING 15.2cm I.D. (6")
HOLE DIAMETER 21.6cm(8-1/2")
HOOK-WALL PACKER 15.2cm I.D. (6")
53.3 m (175')
HOLE DIAMETER 21.6cm(8-1/2")
153.0 m (502')
ELEVATION 584m (1916.3')
0-10.7 m - SALTSBURG SANDSTONE; LIGHT BROWN TO SLIGHTLY REDDISH BROWN, QUARTOSE, MODERATELY HARD TO VERY HARD; (0-35') WEATHERED NEAR SURFACE, SOME LOCAL MODERATELY WEATHERED, OTHERWISE FRESH; FEW INTERBEDS OF SHALE AND SHALEY SANDSTONE; EROSIONAL LOWER CONTACT; WATER ENCOUNTERED AT 4.6 m (15').
10.7-24.4 m - SHALE, LIMESTONE; GRAY LIMESTONE AND LIMEY SHALE BOUNDED BY UPPER AND LOWER SHALE UNITS. UPPER (35-80') SHALE: MODERATE BROWN AND GRAY, MODERATELY HARD; INTERBEDS OF LIGHT BROWN VERY FINE GRAINED SANDSTONE. LOWER SHALE: GRAY AND RED BROWN TO MAROON (MEYERSDALE RED BED?); MINOR GRAY CLAY ASSOCIATED; INCREASE IN FLOW AT 17.7 m (58').
24.4-29.9 m - BUFFALO SANDSTONE-SILTSTONE; GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO (80-98') VERY HARD.
29.9-43.6 m - SHALE; GRAY TO DARK GRAY, MODERATELY HARD; SOME INTERBEDDED SILTSTONE AND SANDSTONE; INCREASE IN (98-143') FLOW AT 38 m (118'), TOTAL FLOW 0.3 l s (5 gpm).
43.6-49.4 m - BRUSH CREEK MARINE SHALE; DARK GRAY, MODERATELY HARD; BLACK ORGANIC MATERIAL COMMON. (143-162')
49.4-49.7 m (162-163') - BRUSH CREEK COAL
49.7-60.0 m - CORINTH SANDSTONE; GRAY TO LIGHT GRAY, VERY FINE TO FINE GRAINED, HARD TO VERY HARD; MINOR (163-197') SHALE AT UPPER CONTACT.
60.0-72.2 m - SHALE-SILTSTONE; GRAY TO GRAY BROWN AND LIGHT TO MODERATE GRAY, MODERATELY HARD, FISSILE; (197-237') INTERBEDS OF VERY FINE GRAINED SANDSTONE. RED SANDSTONE-SHALE NEAR MIDDLE OF UNIT (MAROON RED BED?), VERY HARD.
72.2-80.2 m - MAHONING SANDSTONE, VERY LIGHT GRAY TO LIGHT BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, VERY (237-263') HARD.
80.2-85.6 m - UPPER FREEPORT LIMESTONE; INTERBEDS OF LIMESTONE AND LIMEY SHALE, BROWN TO GRAY BROWN, MODER- (263-281') ATELY HARD TO HARD.
85.6-88.1 m - CLAY; GRAY. (281-289')
88.1-100.3 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN, VERY FINE GRAINED, (289-329') ARGILLACEOUS, MODERATELY HARD TO VERY HARD.
100.3-101.2 m (329-332') - LOWER FREEPORT "D" COAL.
101.2-112.8 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN, VERY FINE TO FINE GRAINED, (332-370') QUARTOSE, MODERATELY HARD TO HARD; BROWN TO GRAY BROWN CLAY AT TOP OF UNIT; INCREASE IN FLOW AT 100.0 m (328'), TOTAL FLOW 0.5-0.8 l s (8-10 gpm)
112.8-113.4 m (370-372') - UPPER KITTANNING "C PRIME" COAL.
113.4-134.4 m - UPPER WORTHINGTON SANDSTONE; GRAY TO GRAY BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, (372-441') ARGILLACEOUS, MODERATELY HARD TO VERY HARD; INTERBEDS OF SILTSTONE AND SHALE.
134.4-135.0 m (441-443') - MIDDLE KITTANING "C" COAL.
135.0-150.0 m - LOWER WORTHINGTON SANDSTONE; GRAY, VERY FINE GRAINED, QUARTOSE, VERY HARD; MINOR DARK GRAY (443-492') SHALE AT UPPER AND LOWER CONTACTS; INCREASE IN FLOW AT 148.4 m (486').
150.0-151.5 m (492-497') - LOWER KITTANNING "B" COAL.
151.5-153.0 m - SHALE; GRAY TO GRAY BROWN. (497-502')

Figure 18. Drilling log: Dewatering well P-3.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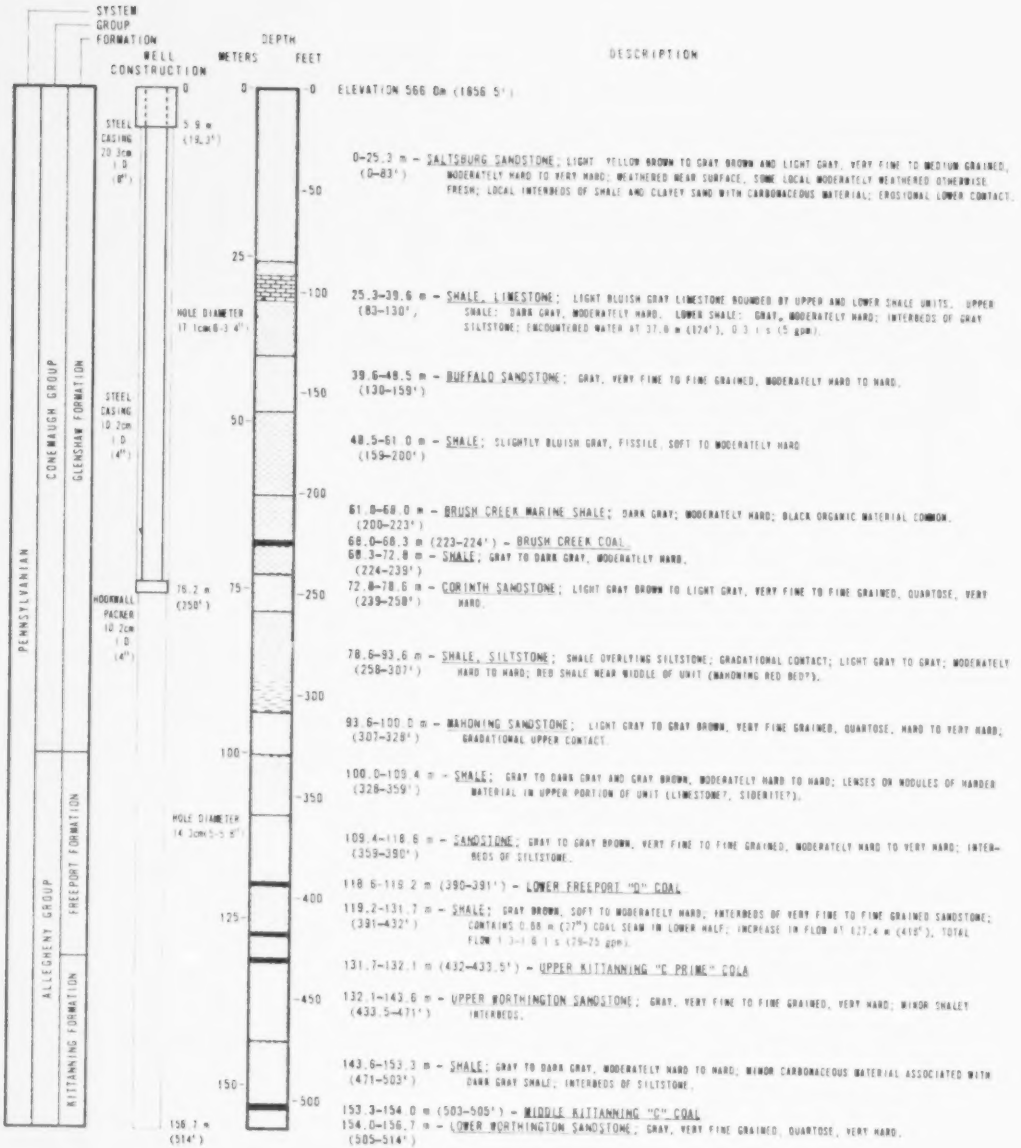
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
QUAT.
PENNSYLVANIAN
CONEMAUGH GROUP
ALLEGHENY GROUP
GLENSHAW FORMATION
FREEPORT FORMATION
KITTANNING FORMATION
STEEL CASING 20.3cm I.D. (8")
8.1 m (26.7')
STEEL CASING 10.2cm I.D. (4")
HOLE DIAMETER (17.1cm(6-3/4")
HOOKNALL PACKER 10.2cm I.D. (4")
51.8 m (170')
HOLE DIAMETER 14.3cm(5-5/8")
147.9 m (485')
ELEVATION 566.3m (1857.8')
0-6.7 m - COLLUVIUM; SANDY CLAYEY SILT, MODERATE BROWN TO YELLOW BROWN. (0-22')
6.7-21.9 m - SHALE; GRAY AND BROWN WITH SOME RED NEAR BASE (MEYERSDALE RED BED?), LOCALLY SANDY, MODERATELY HARD. (22-72')
21.9-26.2 m - BUFFALO SANDSTONE; GRAY TO GREENISH GRAY, VERY FINE TO FINE GRAINED, MICACEOUS, ARGILLACEOUS, MODERATELY HARD TO HARD. (72-86')
26.2-47.5 m - SHALE; GRAY TO GRAY BROWN AND DARK GRAY, SLIGHTLY SANDY, SOME INTERBEDDED SANDSTONE; ENCOUNTERED WATER AT 33.2 m (109'). (86-156')
47.5-48.2 m (156-158') - BRUSH CREEK COAL.
48.2-58.8 m - CORINTH SANDSTONE; LIGHT BROWN TO GRAY, VERY FINE GRAINED, QUARTOSE, VERY HARD. (158-193')
58.8-74.1 m - SHALE; GRAY, SANDY, MODERATELY HARD; INTERBEDS OF SILTSTONE AND VERY FINE GRAINED SANDSTONE. (193-243')
74.1-78.9 m - MAHONING SANDSTONE; LIGHT GRAY TO GRAY BROWN, VERY FINE GRAINED, QUARTOSE, VERY HARD; LOCAL INTERBEDS OF GRAY TO GRAY BROWN SHALE. (243-259')
78.9-84.7 m - UPPER FREEPORT LIMESTONE; INTERBEDS OF LIMESTONE AND LIMEY SHALE, GRAY; MINOR INTERBEDS OF DARK GRAY SANDY SHALE. (259-278')
84.7-98.5 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; CONTAINS POSSIBLE THIN COAL SEAM ABOVE BASE; INCREASE IN FLOW AT 98.0 M (315'), TOTAL FLOW 1.5 l/s (20 gpm). (278-323')
98.5-99.7 m (323-327') - LOWER FREEPORT "D" COAL.
99.7-111.6 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN AND GREENISH GRAY, VERY FINE TO FINE GRAINED, MODERATELY HARD TO VERY HARD. (327-365')
111.6-112.5 m (366-369') - UPPER KITTANNING "C PRIME" COAL.
112.5-132.9 m - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN, SANDY, VERY FINE TO FINE GRAINED, MODERATELY HARD TO VERY HARD. (369-436')
132.9-133.5 m (436-438') - MIDDLE KITTANNING "C" COAL.
133.5-147.6 m - LOWER WORTHINGTON SANDSTONE; GRAY, VERY FINE GRAINED, QUARTOSE, VERY HARD; MINOR GRAY SHALE NEAR BASE; INCREASE IN FLOW AT 147.2 m (483'), TOTAL FLOW 0.2 l/s (30 gpm). (438-485')

Figure 19. Drilling log: Dewatering well P-4.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GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
PENNSYLVANIAN
CONEMAUGH GROUP
GLENSHAW FORMATION
ALLEGHENY GROUP
FREEPORT FORMATION
KITTANNING FORMATION
STEEL CASING 20.3cm I.D (8")
5.9 m (19.3')
HOLE DIAMETER 17.1cm (6.3/4")
STEEL CASING 10.2cm I.D (4")
HOOKBALL PACKER 10.2cm I.D (4")
156.7 m (514')
HOLE DIAMETER 14.3cm (5.5/8")
ELEVATION 566.0m (1856.5')
0-25.3 m - SALTSBURG SANDSTONE; LIGHT YELLOW BROWN TO GRAY BROWN AND LIGHT GRAY, VERY FINE TO MEDIUM GRAINED, MODERATELY HARD TO VERY HARD; WEATHERED NEAR SURFACE, SOME LOCAL MODERATELY WEATHERED OTHERWISE FRESH; LOCAL INTERBEDS OF SHALE AND CLAYEY SAND WITH CARBONACEOUS MATERIAL; EROSIONAL LOWER CONTACT.
25.3-39.6 m - SHALE, LIMESTONE; LIGHT BLUISH GRAY LIMESTONE BOUNDED BY UPPER AND LOWER SHALE UNITS. UPPER SHALE: DARK GRAY, MODERATELY HARD. LOWER SHALE: GRAY, MODERATELY HARD; INTERBEDS OF GRAY SILTSTONE; ENCOUNTERED WATER AT 37.0 m (124'), 0.3 l s (5 gpm).
39.6-48.5 m - BUFFALO SANDSTONE; GRAY, VERY FINE TO FINE GRAINED, MODERATELY HARD TO HARD.
48.5-61.0 m - SHALE; SLIGHTLY BLUISH GRAY, FISSILE, SOFT TO MODERATELY HARD
61.0-68.0 m - BRUSH CREEK MARINE SHALE; DARK GRAY; MODERATELY HARD; BLACK ORGANIC MATERIAL COMMON.
68.0-68.3 m (223-224') - BRUSH CREEK COAL
68.3-72.8 m - SHALE; GRAY TO DARK GRAY, MODERATELY HARD.
72.8-78.6 m - CORINTH SANDSTONE; LIGHT GRAY BROWN TO LIGHT GRAY, VERY FINE TO FINE GRAINED, QUARTOSE, VERY HARD.
78.6-93.6 m - SHALE, SILTSTONE; SHALE OVERLYING SILTSTONE; GRADATIONAL CONTACT; LIGHT GRAY TO GRAY; MODERATELY HARD TO HARD; RED SHALE NEAR MIDDLE OF UNIT (MAHONING RED BED?).
93.6-100.0 m - MAHONING SANDSTONE; LIGHT GRAY TO GRAY BROWN, VERY FINE GRAINED, QUARTOSE, HARD TO VERY HARD; GRADATIONAL UPPER CONTACT.
100.0-109.4 m - SHALE; GRAY TO DARK GRAY AND GRAY BROWN, MODERATELY HARD TO HARD; LENSES OR NODULES OF HARDER MATERIAL IN UPPER PORTION OF UNIT (LIMESTONE?, SIDERITE?).
109.4-118.6 m - SANDSTONE; GRAY TO GRAY BROWN, VERY FINE TO FINE GRAINED, MODERATELY HARD TO VERY HARD; INTERBEDS OF SILTSTONE.
118.6-119.2 m (390-391') - LOWER FREEPORT "D" COAL
119.2-131.7 m - SHALE; GRAY BROWN, SOFT TO MODERATELY HARD; INTERBEDS OF VERY FINE TO FINE GRAINED SANDSTONE; CONTAINS 0.68 m (27") COAL SEAM IN LOWER HALF; INCREASE IN FLOW AT 127.4 m (418'), TOTAL FLOW 1.3-1.6 l s (75-25 gpm).
131.7-132.1 m (432-433.5') - UPPER KITTANNING "C PRIME" COAL
132.1-143.6 m - UPPER WORTHINGTON SANDSTONE; GRAY, VERY FINE TO FINE GRAINED, VERY HARD; MINOR SHALEY INTERBEDS.
143.6-153.3 m - SHALE; GRAY TO DARK GRAY, MODERATELY HARD TO HARD; MINOR CARBONACEOUS MATERIAL ASSOCIATED WITH DARK GRAY SHALE; INTERBEDS OF SILTSTONE.
153.3-154.0 m (503-505') - MIDDLE KITTANNING "C" COAL
154.0-156.7 m - LOWER WORTHINGTON SANDSTONE; GRAY, VERY FINE GRAINED, QUARTOSE, VERY HARD.

Figure 20. Drilling log: Observation well OB-1.

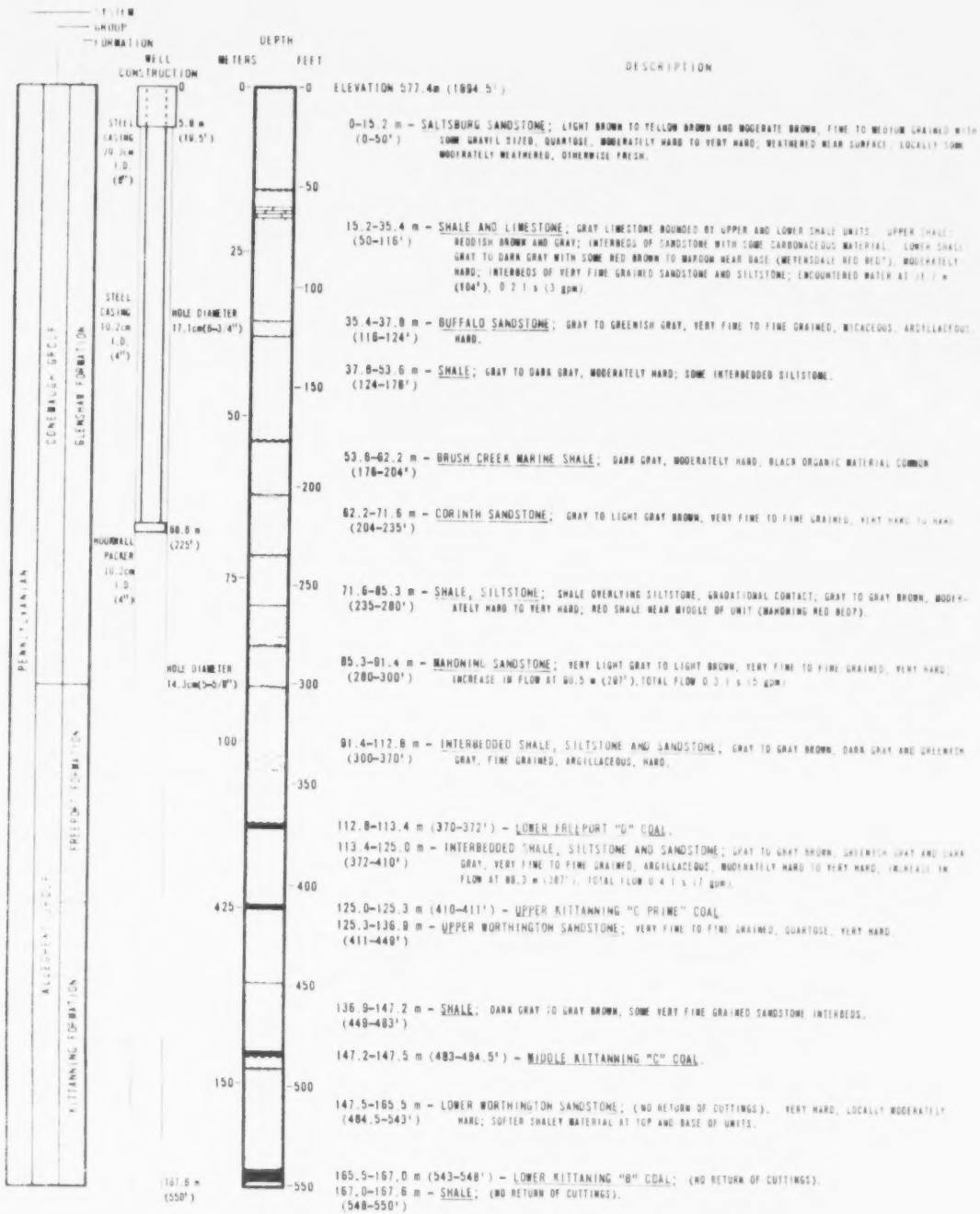

SYSTEM
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
PENN. SYSTEM
CONEMAUGH GROUP
GLENCHAR FORMATION
ALLEGHENY GROUP
FREEPORT FORMATION
KITTANNING FORMATION
STEEL CASING 20.3cm I.D. (8")
5.8 m (19.5')
STEEL CASING 10.2cm I.D. (4")
HOLE DIAMETER 17.1cm(6-3/4")
HOOKBALL PACKER 10.2cm I.D. (4")
68.8 m (225')
HOLE DIAMETER 14.3cm(5-5/8")
167.6 m (550')
0
25
50
75
100
125
150
0
50
100
150
200
250
300
350
400
450
500
550
ELEVATION 577.4m (1894.5')
0-15.2 m (0-50') - SALTSBURG SANDSTONE; LIGHT BROWN TO YELLOW BROWN AND MODERATE BROWN, FINE TO MEDIUM GRAINED WITH SOME GRAVEL SIZED, QUARTOSE, MODERATELY HARD TO VERY HARD; WEATHERED NEAR SURFACE, LOCALLY SOME MODERATELY WEATHERED, OTHERWISE FRESH.
15.2-35.4 m (50-116') - SHALE AND LIMESTONE; GRAY LIMESTONE BOUNDED BY UPPER AND LOWER SHALE UNITS. UPPER SHALE: REDDISH BROWN AND GRAY; INTERBEDS OF SANDSTONE WITH SOME CARBONACEOUS MATERIAL. LOWER SHALE: GRAY TO DARK GRAY WITH SOME RED BROWN TO MAROON NEAR BASE (WESTERNDALE RED BED?), MODERATELY HARD; INTERBEDS OF VERY FINE GRAINED SANDSTONE AND SILTSTONE; ENCOUNTERED WATER AT 35.7 m (104'), 0.2 l/s (3 gpm).
35.4-37.8 m (116-124') - BUFFALO SANDSTONE; GRAY TO GREENISH GRAY, VERY FINE TO FINE GRAINED, MICACEOUS, ARGILLACEOUS, HARD.
37.8-53.6 m (124-176') - SHALE; GRAY TO DARK GRAY, MODERATELY HARD; SOME INTERBEDDED SILTSTONE.
53.6-62.2 m (176-204') - BRUSH CREEK MARINE SHALE; DARK GRAY, MODERATELY HARD, BLACK ORGANIC MATERIAL COMMON
62.2-71.6 m (204-235') - CORINTH SANDSTONE; GRAY TO LIGHT GRAY BROWN, VERY FINE TO FINE GRAINED, VERY HARD TO HARD.
71.6-85.3 m (235-280') - SHALE, SILTSTONE; SHALE OVERLYING SILTSTONE, GRADATIONAL CONTACT; GRAY TO GRAY BROWN, MODERATELY HARD TO VERY HARD; RED SHALE NEAR MIDDLE OF UNIT (MAHONING RED BED?).
85.3-91.4 m (280-300') - MAHONING SANDSTONE; VERY LIGHT GRAY TO LIGHT BROWN, VERY FINE TO FINE GRAINED, VERY HARD; INCREASE IN FLOW AT 90.5 m (297'), TOTAL FLOW 0.3 l/s (5 gpm).
91.4-112.8 m (300-370') - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN, DARK GRAY AND GREENISH GRAY, FINE GRAINED, ARGILLACEOUS, HARD.
112.8-113.4 m (370-372') - LOWER FREEPORT "D" COAL.
113.4-125.0 m (372-410') - INTERBEDDED SHALE, SILTSTONE AND SANDSTONE; GRAY TO GRAY BROWN, GREENISH GRAY AND DARK GRAY, VERY FINE TO FINE GRAINED, ARGILLACEOUS, MODERATELY HARD TO VERY HARD; INCREASE IN FLOW AT 88.3 m (287'), TOTAL FLOW 0.4 l/s (7 gpm).
125.0-125.3 m (410-411') - UPPER KITTANNING "C PRIME" COAL.
125.3-136.9 m (411-449') - UPPER WORTHINGTON SANDSTONE; VERY FINE TO FINE GRAINED, QUARTOSE, VERY HARD.
136.9-147.2 m (449-483') - SHALE; DARK GRAY TO GRAY BROWN, SOME VERY FINE GRAINED SANDSTONE INTERBEDS.
147.2-147.5 m (483-484.5') - MIDDLE KITTANNING "C" COAL.
147.5-165.5 m (484.5-543') - LOWER WORTHINGTON SANDSTONE; (NO RETURN OF CUTTINGS). VERY HARD, LOCALLY MODERATELY HARD; SOFTER SHALEY MATERIAL AT TOP AND BASE OF UNITS.
165.5-167.0 m (543-548') - LOWER KITTANING "B" COAL; (NO RETURN OF CUTTINGS).
167.0-167.6 m (548-550') - SHALE; (NO RETURN OF CUTTINGS).

Figure 21. Drilling log: Observation well OB-2.

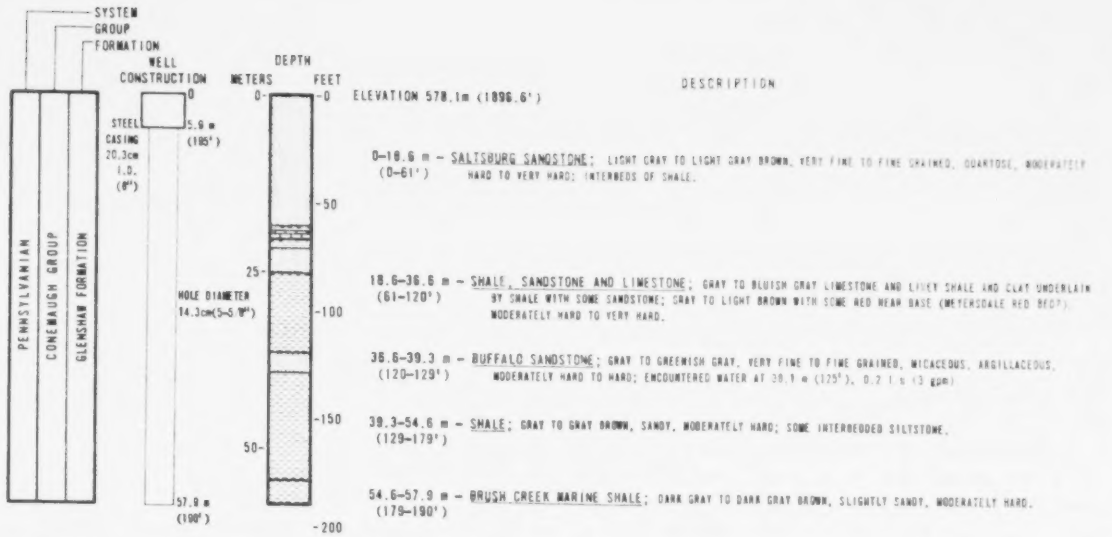

SYSTEM
GROUP
FORMATION
WELL CONSTRUCTION
DEPTH
METERS
FEET
DESCRIPTION
ELEVATION 578.1m (1896.6')
PENNSYLVANIAN
CONEMAUGH GROUP
GLENSHAW FORMATION
STEEL CASING 20.3cm I.D. (8")
5.9 m (195')
HOLE DIAMETER 14.3cm(5-5 8")
57.9 m (190')
0-18.6 m (0-61') - SALTSBURG SANDSTONE; LIGHT GRAY TO LIGHT GRAY BROWN, VERY FINE TO FINE GRAINED, QUARTOSE, MODERATELY HARD TO VERY HARD; INTERBEDS OF SHALE.
18.6-36.6 m (61-120') - SHALE, SANDSTONE AND LIMESTONE; GRAY TO BLUISH GRAY LIMESTONE AND LIKEY SHALE AND CLAY UNDERLAIN BY SHALE WITH SOME SANDSTONE; GRAY TO LIGHT BROWN WITH SOME RED NEAR BASE (METERSDALE RED BED?). MODERATELY HARD TO VERY HARD.
36.6-39.3 m (120-129') - BUFFALO SANDSTONE; GRAY TO GREENISH GRAY, VERY FINE TO FINE GRAINED, MICACEOUS, ARGILLACEOUS, MODERATELY HARD TO HARD; ENCOUNTERED WATER AT 38.1 m (125'), 0.2 l/s (3 gpm)
39.3-54.6 m (129-179') - SHALE; GRAY TO GRAY BROWN, SANDY, MODERATELY HARD; SOME INTERBEDDED SILTSTONE.
54.6-57.9 m (179-190') - BRUSH CREEK MARINE SHALE; DARK GRAY TO DARK GRAY BROWN, SLIGHTLY SANDY, MODERATELY HARD.
0
-50
25-
-100
-150
50-
-200

Figure 22. Drilling log: Observation well OB-3.

SECTION 7

PILOT DEWATERING PROGRAM

METHOD OF APPROACH

The objectives of the dewatering program were to intercept a major portion of the groundwater inflow to the Main G study area and thereafter to analyze the cost-effectiveness of mine dewatering as a means of controlling AMD. The approach used was to construct vertical wells adjacent to the mine openings to intercept groundwater before it could be degraded in the mine. The wells were located along a line parallel to the mine face in an attempt to create a hydraulic drawdown barrier against groundwater movement toward the underground study area. Despite the complexities of the groundwater flow system (described in Section 6), the results of the pilot program demonstrated that well dewatering can be effective and also pointed out some of the practical problems involved. The program further demonstrated that the untreated water pumped from a dewatering system can be of significantly higher quality than treated mine water and can be discharged directly into a receiving stream.

Pumped wells were used for this study because they provide more information, and create less disruption of mining and less safety hazard than other possible methods. A discussion of relevant alternative dewatering methods is presented in Section 8.

DEVELOPMENT

The program, which was used initially to develop baseline data, as well as to prepare for the dewatering program, was carried out in three phases. The initial phase consisted of the development of an underground monitoring facility. In the second phase surface geologic mapping was performed and one test or pumping well and two observation wells were constructed. The purpose of this phase was to determine the feasibility of a dewatering scheme using wells, and to provide the necessary data for locating additional test and observation wells. This phase was completed by the construction and pump testing of these additional wells for the pilot dewatering program. The third and final phase was the selection of appropriate pumps and generator. These activities were detailed in the preceding section.

OPERATION

Based on data obtained during the preparation phases of this study, the pilot well dewatering program was scheduled to be operated for two or three weeks. During operation, monitoring and sampling on the surface and in the mine were carried out on a continuous basis. Equipment for both of these

operations was the same as that used during the pump testing. Drainage from the three dewatering wells was discharged into a small basin located near P-3 and allowed to flow by gravity through a small culvert under the railroad embankment and down to the nearby stream. To prevent possible seepage losses into the subsurface, the basin adjacent to the culvert was plastic lined.

The pilot well dewatering program began with continuous pumping on July 13, 1977. Total pumpage for all three wells initially was only 2.87 ℓ/s (45.5 gpm); Well P-2 was yielding only about 0.2 ℓ/s (3 gpm) and the pumps on P-1 and P-3 were producing only about half their capacity. The effect on mine inflow was only slightly greater than that seen in the individual pump testing of the wells. Pumps were shut off on July 16, and the wells were allowed to recover. Well P-2 was abandoned as a pumping well and the 3.7 kW (5 hp) pump in P-2 was pulled and reinstalled in P-4, which had been used as an observation well (OB-4) up to that point. Pumping resumed on July 18, and the total pumpage was increased to 4.6 ℓ/s (73.5 gpm). Pumping continued at this rate until July 20, when once again the pumps were shut off. This time the cause of shutdown was intense rainfall during the nights of July 19 and 20, accompanied by widespread surface flooding. As a consequence of this storm the underground workings of the Lancashire No. 20 mine were completely inundated. Pumping at the wells and all other field operations was temporarily shut down until the mine could be pumped out. Total rainfall in the area from the storm was greater than 30 cm (12 in.) in a 24-hour period. Water levels in wells started rising before the mine was flooded, indicating recharge from direct surface infiltration of rainfall.

Field operations resumed in late August and the pilot well dewatering program was again started on September 16. Prior to the start of pumping, wells were cleaned to flush out any fine-grained sediments which might have flowed into the wells during the storm. In the mine the underground transport of water was also reorganized following the clean-up of the mine, and monitoring equipment underground was relocated to compensate for this change. With the continuation of pumping in mid-September, minor shake-down problems occurred during the first couple of days of pumping. Pumps were shut down twice because of minor problems with the generator and pump control box. After the problems were corrected, pumping was continuous from September 23 until October 7, 1977.

MONITORING

Prior to the start of dewatering, procedures were determined for monitoring of well discharges and mine inflows and for sampling of both surface and underground waters for water quality monitoring. Equipment and methods used for these operations were similar to those employed during the pump testing and baseline data collection periods.

Flow Measurements

Flow measurements were made on a continuous basis for the inflows to the Main G study area and for discharges of dewatering wells, along with water level measurements in observation wells and pumped wells. Stevens continuous float-operated water-level recorders were installed in the observation wells

to monitor changes in water level. Water levels in pumping wells were measured on a regular basis with an electric water level indicator similar to that used during pump tests. Pumpage rates in the dewatering wells were recorded hourly, initially using a 5 cm (2 in.) digital flow meter. These were removed 3 days after the start of pumping because of jamming from silt and sand sized rock particles. A 20 ℓ (5 gal) bucket, and later a 210 ℓ (55 gal) drum, were used to measure the discharge from each well. This method of flow measurement was considered to be adequate because the discharges were relatively constant with only gradual changes, and the discharge from each well was only 1.3 to 2.5 ℓ/s (20 to 40 gpm). Errors in measurement probably were less than 10 percent, which is adequate for this study.

The underground monitoring system described earlier was utilized to obtain mine inflow data for the section of the mine being studied. Water meter readings were taken once each shift (three times per day) by supervisory personnel working in the Main G area. Meter readings were recorded each shift and then transferred to a master data sheet in the foreman's office each day. These flow measurements were collected daily by Wahler or ENCOTEC personnel. Occasionally personnel from the study team would go underground and take measurements to check the readings being taken by the mine operators.

The Stevens recorder was capable of eight days' unattended data recording. Each week a member of the study team went underground to pick up the recorded data and reset the Stevens recorder. Other equipment was also checked at this time along with a general inspection of the test area of the mine. As described previously, the monitoring system in the Main G study area had to be rearranged after the July 19-20 storm. However, the same monitoring schedule was used when dewatering operations were resumed in September.

Water Quality Monitoring

During the pilot dewatering operation, water samples for analysis were collected from the Main G study area, the dewatering wells and from two stations on Laurel Lick Run, above and below the discharge of the wells, as shown on Figure 23. Grab samples of the creek water were collected 7 times during the September dewatering period spaced at 2 to 4 day intervals.

Grab samples were collected from each pumping well throughout both dewatering periods in July and September. Analyses of these samples provided data to detect changes in water quality in any of the wells during the dewatering periods. Also, these data indicated the acceptability of the well water for direct discharge to the creek according to the discharge requirements of the Commonwealth of Pennsylvania.

Water in the Main G study area was sampled in July from the discharge pipe from the L-2 sump, and in September the sampling station was moved to sample discharge from the L-2 pool as described in Section 6. Water samples at both locations were obtained using the Protech composite sampler. The sampler was set to collect a fixed-volume sample each hour and a 24-hour composite sample was generated. Each day during the dewatering period (including the recovery period), mine personnel would change sample containers and bring the sample to the surface. Occasional grab samples were

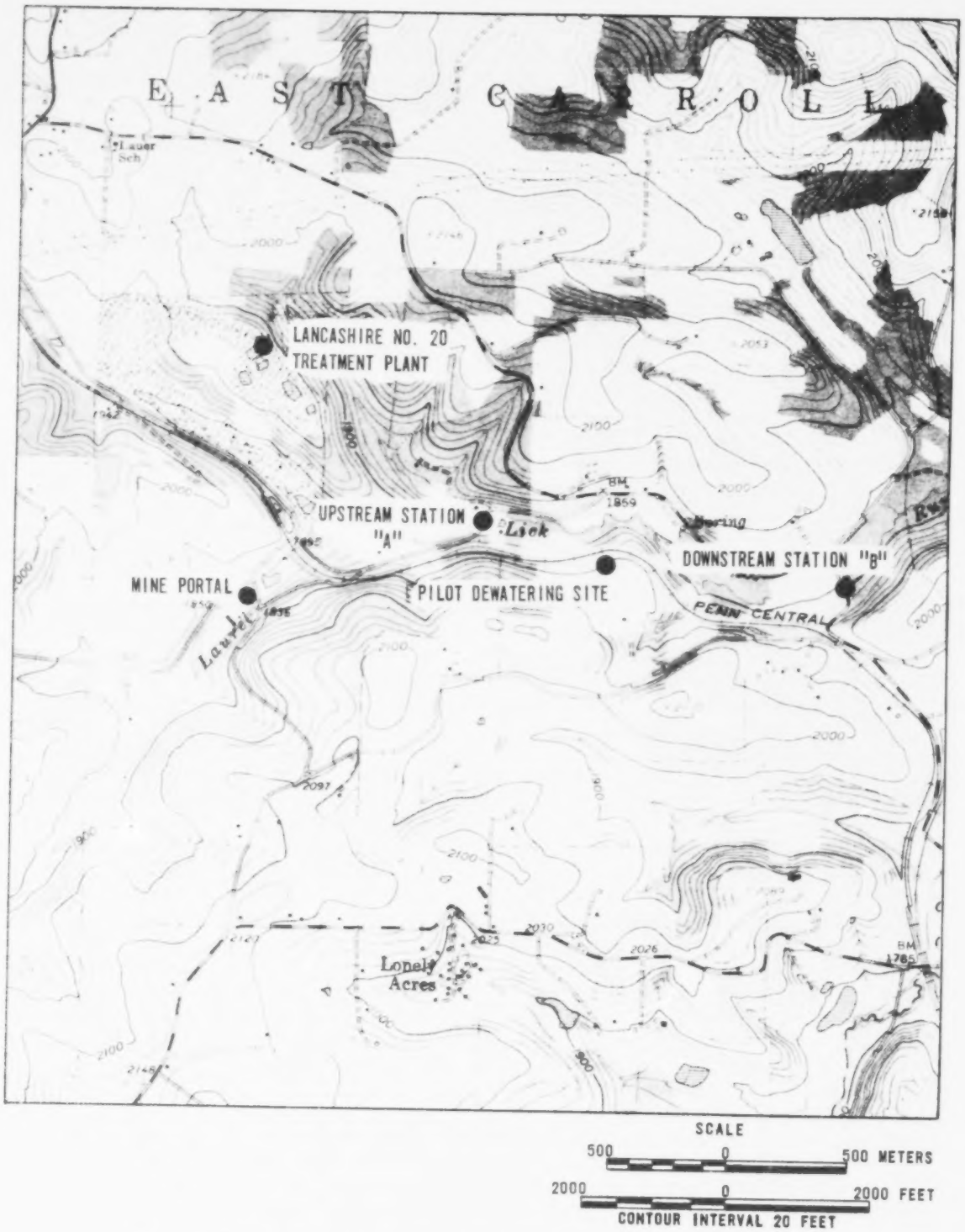


Figure 23. Location map: Surface water sampling stations.

collected to check the results of the composite samples. In addition, grab samples of the F-14 pipeline to the treatment plant were obtained during both dewatering periods. These samples provided analyses to compare with baseline data.

RESULTS

Effect of Pumping on Mine Inflows

The first pilot dewatering started on July 13, 1977, pumping from Wells P-1, P-2, and P-3. The yield of the wells was increased gradually to a maximum total of 2.9 ℓ/s (45.5 gpm). This yield was disappointing because P-1 and P-3 were both expected to yield 2.5 to 3.2 ℓ/s (40 to 50 gpm) each, based on previous pump tests. Well P-2 was not expected to produce much water, as it did not, and the pump was later removed. Pumping continued for about 3 days and was stopped on July 16. During this period flows in the Main G study area of the mine had decreased from 6.7 ℓ/s (107 gpm) to 6.0 ℓ/s (96 gpm), or 10 percent, as shown on Figure 24. About 24 percent of the water pumped from wells was intercepted or diverted from the mine. These results were improved (Figure 25) by changing equipment.

The 3.7 kW (5 hp) pump was removed from P-2 and installed in OB-4, thereafter designated P-4. Pumping resumed on July 18, and total pumpage was increased to 4.64 ℓ/s (73.5 gpm). On July 19, flow in the Main G study area, which had increased to 7.0 ℓ/s (111 gpm) just prior to starting the pumps, rapidly decreased 28 percent to about 5.1 ℓ/s (80 gpm). This indicated that 42 percent of the water pumped from the wells was diverted from mine inflows. The operation was interrupted by mine flooding, but these results were encouraging because the percentage of water intercepted should increase with time.

Dewatering was again resumed on September 16, 1977 after cleaning the wells and resetting pumps. Minor problems with the generator caused two breaks in pumping between September 16 and September 23. After this time the wells were pumped continuously until October 7, 1977. The average total pumpage was approximately 5.2 ℓ/s (82 gpm), varying from about 6.0 ℓ/s (95 gpm) near the start and decreasing to about 4.7 ℓ/s (75 gpm) at the end of the pumping period. The pumping rates for each well are shown on Figure 26; total surface pumpage is shown on Figure 27; and water levels on Figure 28. Water levels in observation wells showed a mixed response to pumping, as shown on Figures 29 through 32. Well OB-1 responded as expected, with generally consistent declines during pumping except near the end of the period. Well OB-2 declined rapidly at first and then more or less stabilized. Water levels in P-2 declined slightly and then rose. Well OB-3 had essentially no response to pumping.

Inflows to the Main G study area (Figure 25) decreased rapidly at first but showed a slight decreasing trend after September 25. Inflows ranged between 4.4 and 4.8 ℓ/s (70 to 76 gpm) after this date and quickly recovered to 7.1 ℓ/s (112 gpm) after pumping stopped. The average inflow rate from September 25 through October 6 was 4.62 ℓ/s (73.2 gpm) with a standard deviation of $\pm$0.12 ℓ/s ($\pm$1.92 gpm). The post-pumping monitoring period, after inflows had recovered, from October 8 through October 22, showed an average

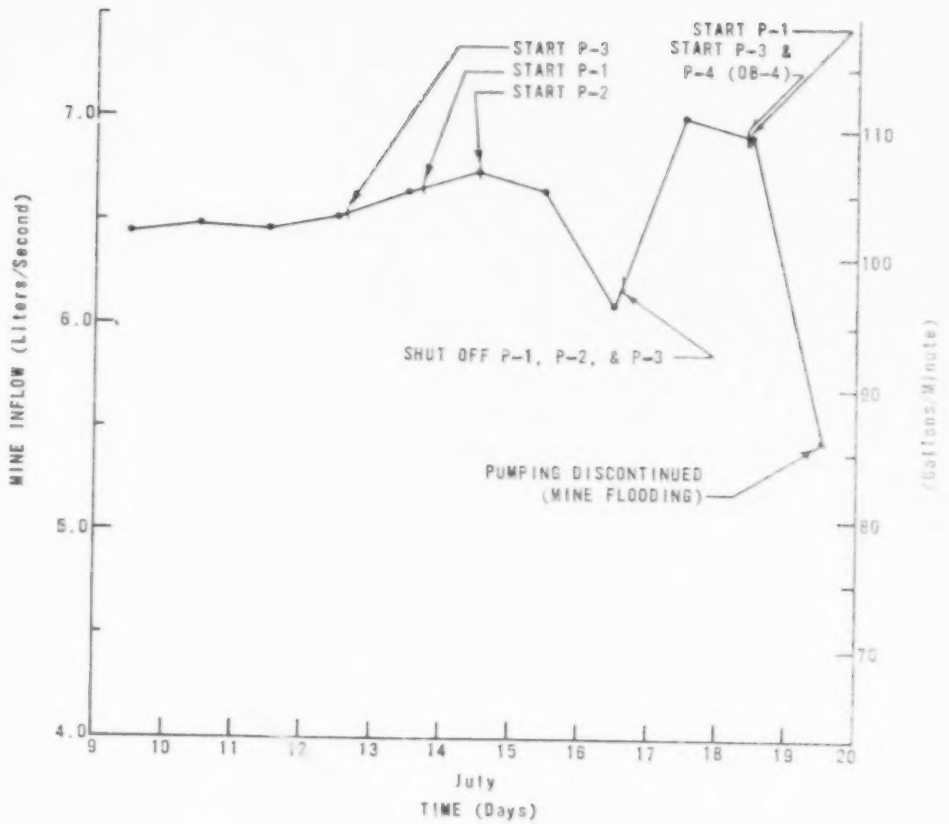


Figure 24. Average daily mine inflow - July 1977 dewatering.

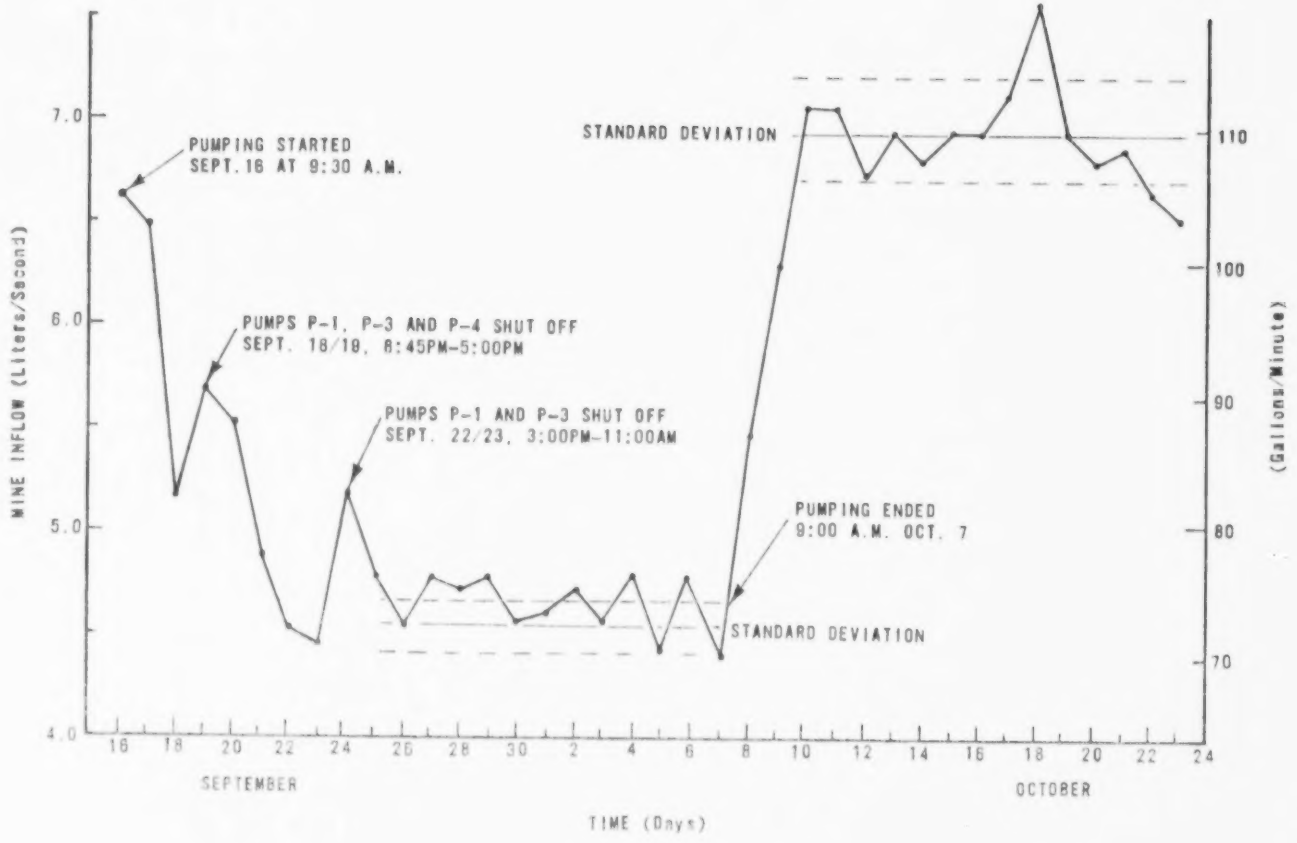


Figure 25. Average daily mine inflow - September 1977 dewatering.

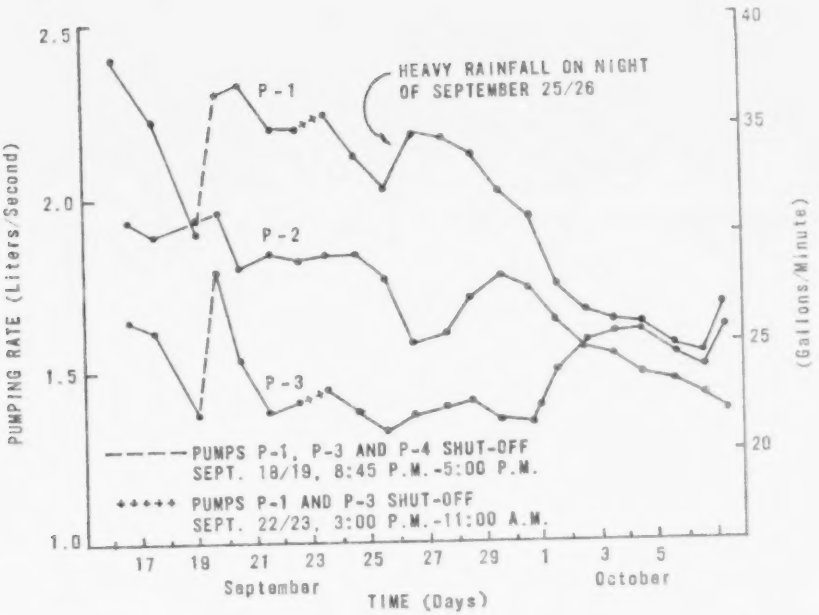


Figure 26. Average daily pumping rates - September 1977 dewatering.

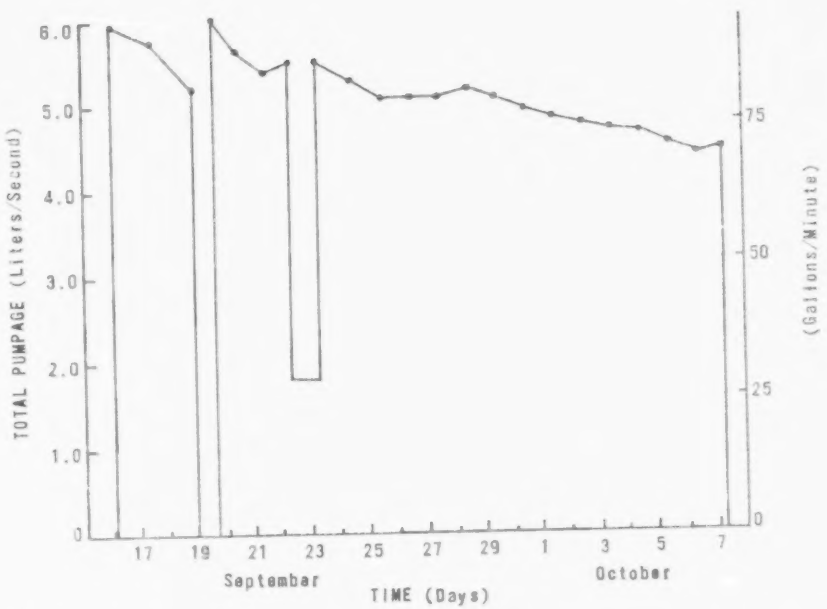


Figure 27. Combined daily pumpage rate - September 1977 dewatering.

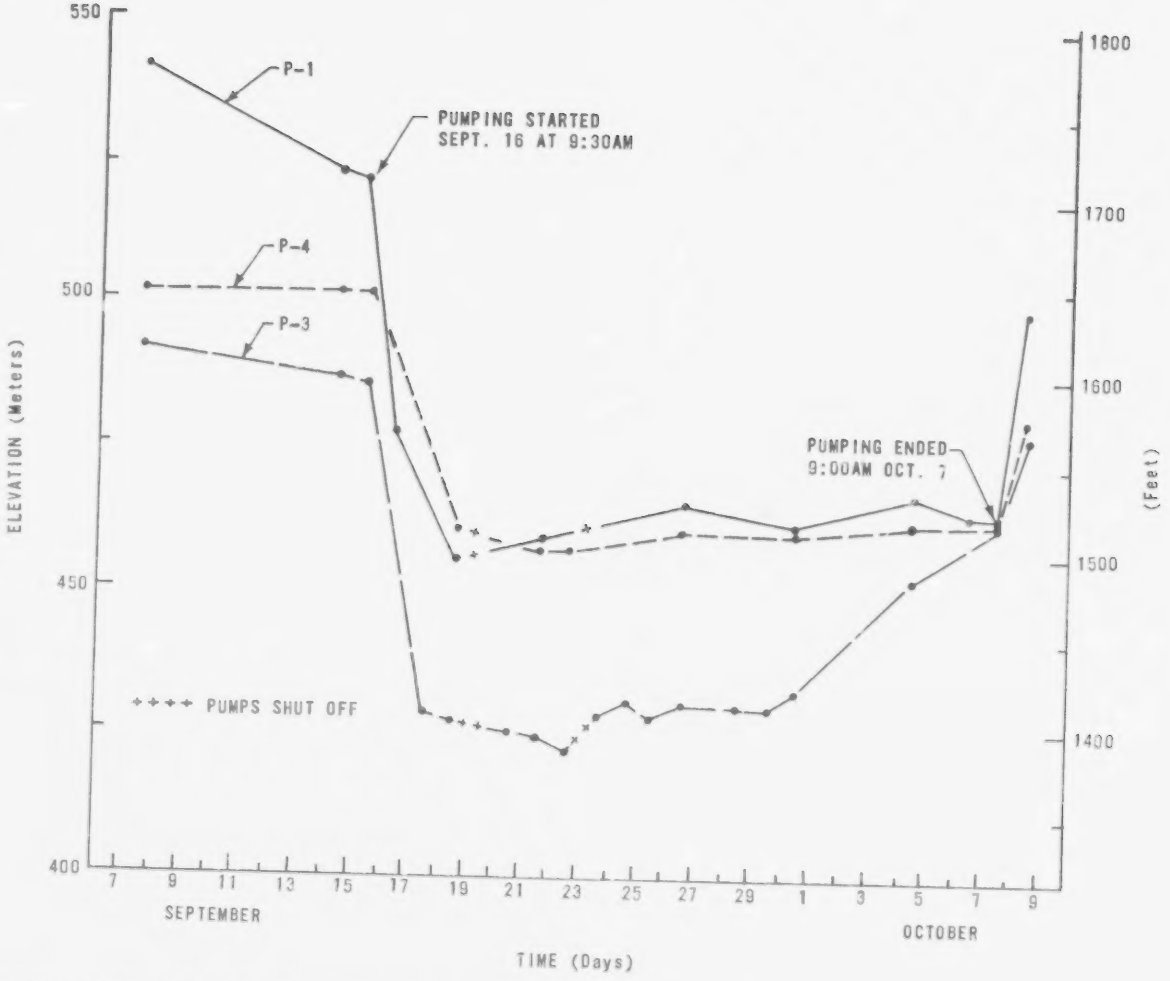


Figure 28. Water level elevations in dewatering wells - September 1977 dewatering period.

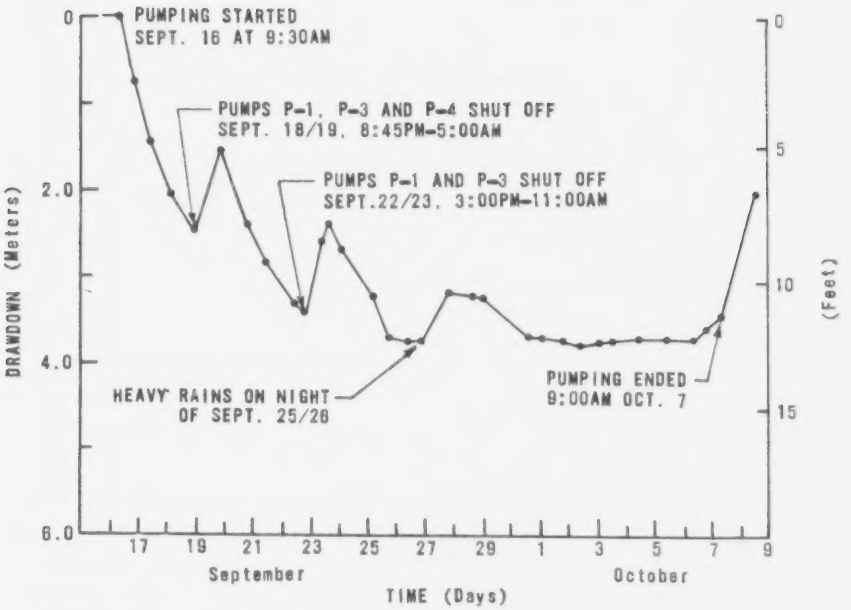


Figure 29. Drawdown: Observation well OB-1 - September 1977 dewatering.

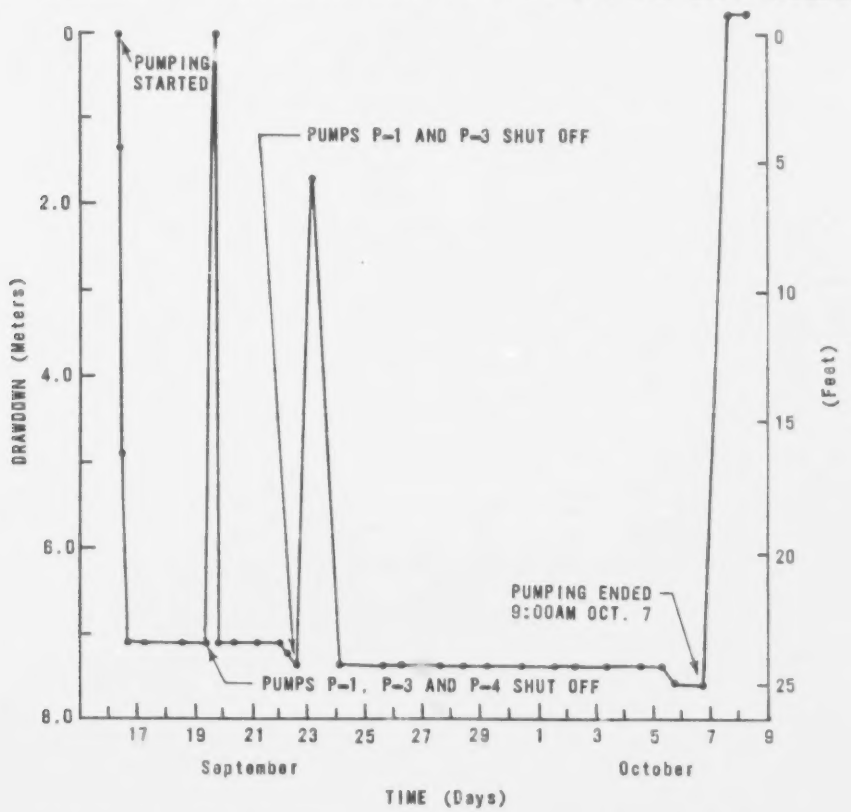


Figure 30. Drawdown: Observation well OB-2 - September 1977 dewatering.

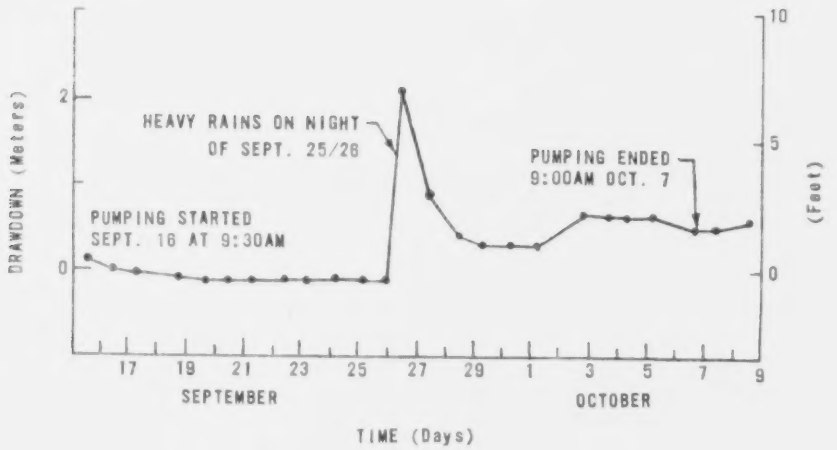


Figure 31. Drawdown: Observation well OB-3 - September 1977 dewatering.

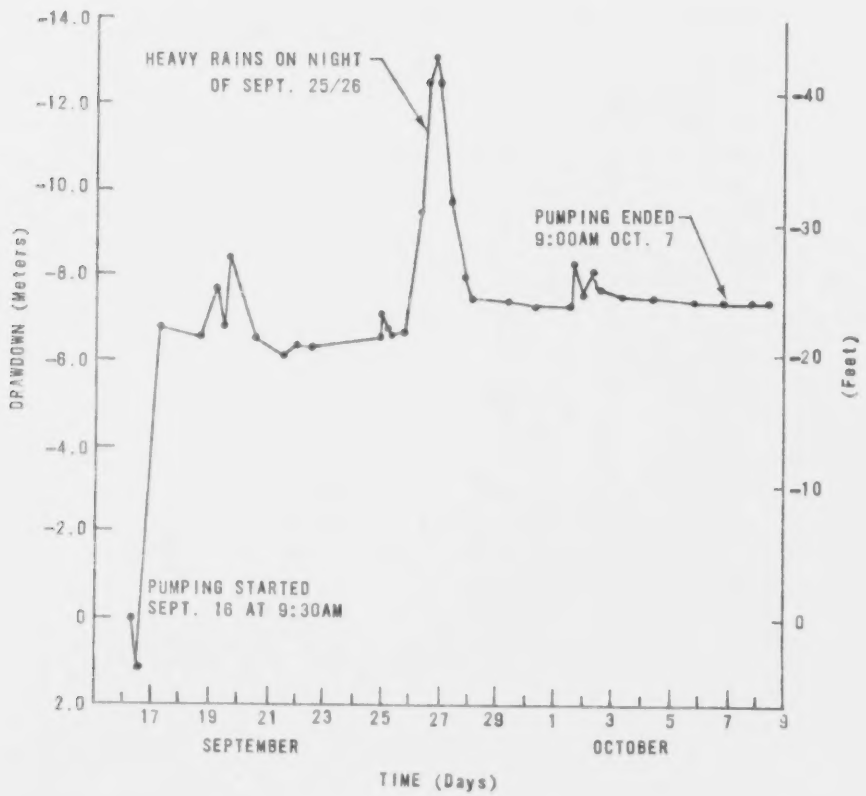


Figure 32. Drawdown: Observation well P-2 - September 1977 dewatering.

flow of 6.9 ℓ/s (110 gpm) with a standard deviation of ±0.25 ℓ/s (±4 gpm). These values are quite comparable with all of the other background data, which showed an average inflow rate of 7.1 ℓ/s (112 gpm), ±0.59 ℓ/s (±9.4 gpm).

As can be seen in Figures 26 through 32, all the pumping and observation wells, with the exception of OB-2, show the effects of recharge from direct precipitation. The pumping and inflow data indicate that the mine study area inflow was decreased by about 34 percent based on average flow rates. It may have been as much as 38 percent, based on measurements at the end of the pumping period and the first inflow measurements after full recovery. These data indicated that 45 percent of the water pumped from wells comprised intercepted or diverted mine inflow, based on average rates for the pumping period. Theoretically, this percentage should increase with time, and at the end of the test up to 56 percent of the water pumped was diverted from mine inflows. Projection of these data on the graph developed by Theis and Conover (1963)[10] to 120 days of pumping indicates that up to 80 percent of the water pumped would be diverted from mine inflows. Recharge events from storms or recharge from streams would reduce these percentages. Therefore for purposes of estimating full scale mine dewatering, it was assumed that 50 to 80 percent of the water yield from dewatering wells would be diverted from mine inflows.

During the dewatering operation, well yields were still disappointing. Only Well P-3 was close to the maximum available drawdown, and that could not be sustained during the latter part of the pumping period. Well P-1 had more than 30 m (100 ft) of available additional drawdown. The 5.6 kW (7.5 hp) pump in each of these two wells was not able to produce at its rated capacity, which was in excess of 2.5 ℓ/s (40 gpm) with a pump lift of 155 m (510 ft). In contrast, the 3.7 kW (5 hp) pump performed relatively well throughout most of the dewatering program and during earlier pump tests. However, the yield of P-4 gradually decreased along with a small increase in drawdown. The impellers on the 5.6 kW (7.5 hp) pumps may have been worn, reducing their capacity, and/or gas blocks may have developed to retard well yield. All three pumping wells produced methane gas, which was detected qualitatively but the volume was not measured. The pumped water obviously had a large amount of gas entrained in the discharge.

Water Quality

A summary of various water quality parameters measured during the two dewatering periods (July and September) is given in Table 5, and water quality data for the two pumping periods are shown in Tables 6, 7, and 8, located at the end of this section.

Water quality was monitored in each of the three pump dewatering wells throughout both dewatering periods. There were some differences between the various wells. The water in Well P-1 was lower in pH, alkalinity, and dissolved solids than the water in the other two wells, while Well P-4 contained water with higher levels of sulfate than the other two wells. Well P-1 water showed a noticeable decrease in calcium, magnesium, and iron levels during the September 1977 pump dewatering period. Calcium and magnesium levels also decreased in Well P-3 water during this period. Total iron, however, was more variable in this well (P-3) and did not exhibit the general downward

trend, although ferrous iron values did decrease. The water from Well P-4 did not show this decreasing trend in the level of certain chemical components and was generally consistent over both the dewatering periods.

Water quality varied somewhat from well to well. The water in Well P-1 was lower in pH, alkalinity, and total dissolved solids when compared to the water in Wells P-3 and P-4. Well P-4 water had the highest pH and dissolved solids along with almost twice the level of sulfate (i.e. 41 mg/ℓ compared to 22 to 24 mg/ℓ). Iron levels in the well water at P-4 were the lowest of the three wells and it was the most similar in character to mine inflows in the underground study area. The variability in water quality between these three wells was probably due to the relative amounts of water produced from the deeper parts of the wells. P-4 probably produced a higher proportion of its water near the "B" coal seam from fractures and from the seam itself. Also, wells with better hydraulic connection with the surface (through fractures) would tend to yield less mineralized water.

The quality of the water from all three dewatering wells was acceptable for direct discharge to a surface stream during the dewatering operation. Iron concentrations during the periods of dewatering were less than 1 mg/ℓ, except during the initial day of pumping. The somewhat elevated iron values during the first day of pumping may have been due to contamination from the casing and other materials in the wells. After the first day, iron concentrations in each of the dewatering wells remained low, although they were somewhat variable. The pH of the well water was acceptable for direct discharge and did not vary significantly during the study. The good quality of well water was an important aspect of the dewatering program. Direct discharge of the intercepted groundwater to the receiving stream is a major benefit of any dewatering scheme. The elimination or reduction in the amount of drainage water to be treated is highly desirable from environmental, operational, and economic perspectives. Water pumped from the dewatering wells was discharged directly to Laurel Lick Run during the dewatering program and caused no significant changes in the water quality of the creek. Laurel Lick Run, which received the discharges from the dewatering wells, was weakly buffered with an average pH of 6.8 during the dewatering operation. Hence the creek water was highly susceptible to fluctuating pH values due to this low alkalinity. On one occasion during the background studies, the pH of the water in the creek dropped to 3.2, indicating the probable release of some acid-containing materials somewhere upstream. Table 7 shows water quality data for Laurel Lick Run at Stations A and B, above and below the dewatering wells.

A comparison of the overall quality of the creek with that of the well waters being discharged during the dewatering program showed that in general the well waters were more stable (i.e. have higher alkalinity) and were lower in sulfate (based on a composite concentration level from the three dewatering wells) than the creek water. Iron values were relatively similar in the well waters and creek, while total dissolved solids were somewhat higher in the well water discharges. With the possible exception of dissolved solids, the well water discharges were of comparable or better quality than the creek water.

The discharge of the well waters would be of some benefit to the creek in terms of pH and alkalinity, as these discharges would add buffering capacity to the creek.

A full scale dewatering program at this mine would produce sufficient quantity of water from the dewatering wells to influence the creek. However, the influence on the quality of the creek water would be positive, since the well water would provide a buffering effect (as described above) and iron concentrations would remain basically the same. The only potentially adverse effect of the dewatering well discharges on the creek water would be a slight increase in dissolved solids. The dissolved solids levels in the water from the dewatering wells range from 124 mg/ℓ to 178 mg/ℓ. These levels are not excessively high and would not cause major water quality problems in the creek.

There are possible uses for water intercepted by dewatering wells other than to discharge it to a surface receiving stream. Assuming good water quality, the intercepted groundwater could possibly be utilized by the mine itself or by nearby farms and communities. These other types of usages (irrigation, water supply, etc.) could provide a direct secondary benefit to those living near mining operations.

During both the July and September dewatering periods, water samples were collected from the L-2 monitoring station. In addition, samples of the roof waters at L-4 were collected during the July period to see if there was any deterioration between the two sampling points. Comparison of the data on samples collected on the same days in July (see Table 8) indicated there was some degradation in sulfates, iron and total dissolved solids. However, the degradation was not substantial, and sampling at L-4 was discontinued. As can be seen in the summary of results (Table 5), iron and sulfate concentrations showed the largest fluctuations throughout the study, with variations of ± 50 percent or more about the mean. Other parameters such as pH, alkalinity, specific conductance, and total dissolved solids were relatively consistent.

The water quality of the inflow to the Main G study area was not affected by the dewatering operation. A comparison of the background data with data collected during the dewatering periods shows virtually no change in any of the parameters monitored. Water quality in the underground study area was good throughout the dewatering program and remained good after the dewatering pumps were shut down.

During the dewatering operations in July and September, additional samples were collected from the F-14 discharge line (influent to the treatment plant). Analyses of these samples are presented on Table 8. The pH and alkalinity values for the F-14 line tended to be higher during the Fall of 1977 than the comparable baseline data collected in the Fall of 1976. Sulfate concentrations did not appear to change appreciably between these periods. Iron values fluctuated considerably from 2.7 mg/ℓ to 18 mg/ℓ during the September/October 1977 survey but showed an overall decrease when compared to 1976 values. Therefore, there was some improvement in quality between the background and dewatering periods. This improvement may have been due to increased flows from the K section of the mine. A longwall panel

encountered high inflows and this water was pumped directly to the F-14 sumps. This movement provided relatively little of the exposure needed for degradation in quality.

The pilot dewatering program did not influence the quality of water pumped to the treatment plant or have an impact on the treatment of the mine water. This was because only a small percentage (5 percent) of the total mine water discharge was affected by the dewatering study. A full-scale dewatering operation would affect the mine water treatment plant in more substantial ways. It would reduce the capacity requirements of the existing water treatment plant and reduce tankage required for mixing and chemical storage as well as the land area required for settling ponds. The quality of the non-intercepted mine drainage water, on an absolute concentration basis, would probably become poorer as drainage from old workings, etc., became a more significant portion of the total treatment plant flow. The total mass flux of pollutants (acid, sulfate, iron, etc.) from the mine would be reduced. This would be due to the decrease in the quantities of water contacting mined areas and subsequently becoming contaminated. Thus, a reduction in chemical usage at the water treatment facility could be expected. This reduction would lessen the amount of chemical sludge produced and the associated problems of sludge disposal.

The benefits of decreased water treatment plant size, together with reductions in chemical usage, extend beyond the obvious economic advantage. Lower chemical usage requirements are advantageous both from an environmental and from an energy viewpoint. Surface receiving streams in the area of dewatering operations would be receiving a more natural and generally better quality water (assuming good quality water from the dewatering wells). Some energy savings would be realized by a reduction in both the production and transportation of the chemicals required at the water treatment plant. Requirements for sludge disposal would be lower, resulting in less land usage and a lowering of the potential pollution hazards involved in handling sludge and supernatant sludge.

TABLE 5. SUMMARY RESULTS - WATER QUALITY

Location	Dewatering Period	pH		Alkalinity mg/ℓ $CaCO_3$		Specific Conductance µS/cm @25°C		Sulfate mg/ℓ		Total Dissolved Solids mg/ℓ		Calcium mg/ℓ		Magnesium mg/ℓ		Ferrous Iron mg/ℓ		Total Iron mg/ℓ	
		Av[a]	SD[b]	Av	SD	Av	SD	Av	SD	Av	SD	Av	SD	Av	SD	Av	SD	Av	SD
WELL WATERS	Baseline	8.3	--	117	--	264	--	19	--	156	--	--	--	--	--	--	--	0.007	--
	July Program																		
	P-1	7.4	--	89	--	195	--	19	--	143	--	--	--	--	--	0.09	--	0.47	--
	P-3	7.7	0.2	108	12	230	16	18	3	147	3	--	--	--	--	--	--	1.37	1.45
	P-4	8.0	--	110	--	--	--	39	--	182	--	--	--	--	--	0.02	--	0.10	--
	Sept. Program																		
	P-1	6.8	0.2	70	15	176	37	22	11	124	23	--	--	--	--	0.24	0.30	0.48	0.39
	P-3	7.3	0.1	104	12	337	21	24	10	161	7	--	--	--	--	0.12	0.18	0.81	0.17
	P-4	7.6	0.2	107	4	267	19	41	3	178	7	--	--	--	--	<0.02	--	0.22	0.20
RECEIVING STREAM	Baseline																		
	Station A	6.98 (6.35	0.4 1.59)[c]	16	6	214	120	100	70	150	26	12	--	4.6	--	0.07	0.07	0.59	0.54
	Sept. Program																		
	Station A	6.84	0.3	15	5	166	51	43	29	126	41	15	--	4.9	1.2	<0.02	--	0.42	0.10
	Station B	6.77	0.3	16	5	146	78	45	21	124	44	14	4	4.5	1.3	<0.02	--	0.37	0.16
UNDERGROUND STATIONS	Baseline																		
	Roof	8.2	0.2	118	9	263	46	31	13	213	21	15	--	4.0	--	--	--	0.42	0.78
	L-2	8.0	0.1	114	6	260	11	50	22	222	10	16	--	4.1	--	--	--	0.09	0.08
	July Program																		
	Roof	8.0	0.3	111	15	300	9	28	9	210	11	--	--	--	--	--	--	0.72	0.46
	L-2	8.0	0.2	122	6	265	9	18	1	175	3	--	--	--	--	--	--	0.04	0.02
	Sept. Program																		
	L-2	7.8	0.1	114	5	265	14	32	3	184	7	19	3	4.0	0.2	--	--	0.35	0.18

[a] Average.

[b] Standard Deviation.

[c] Includes one value of pH 3.2.

TABLE 6. PILOT PROGRAM: WATER QUALITY DATA-WELL WATERS

Location	Date 1977	pH	Alkalinity mg/ℓ $CaCO_3$	Acidity mg/ℓ $CaCO_3$	Specific Conductance μS/cm @25°C	Sulfate mg/ℓ	Total Dissolved Solids mg/ℓ	Calcium mg/ℓ	Magnesium mg/ℓ	Ferrous Iron mg/ℓ	Total Iron mg/ℓ
Well P-1	7/15	7.4	88	--	195	20	152	--	--	0.10	0.58
	7/19	7.3	89	--	--	17	133	--	--	0.07	0.35
	9/10	6.50	58	9	190	48	140	22	5.7	0.85	1.2
	9/17	6.90	74	14	210	30	141	15	3.9	0.55	0.88
	9/18	6.90	80	--	210	25	152	13	3.6	0.42	0.73
	9/20	6.75	62	--	140	18	104	11	3.0	0.17	0.54
	9/22	7.00	80	8	190	16	125	9	2.6	0.08	0.24
	9/24	7.02	84	13	200	17	135	10	2.7	<0.02	0.18
	9/27	6.39	36	3	98	16	78	6.6	2.2	0.02	0.25
	9/30	6.91	78	10	190	15	125	8.9	2.4	<0.02	0.09
	10/4	6.71	74	11	160	17	114	8.9	2.5	<0.02	0.18
Well P-3	7/12	7.5	90	--	212	23	146	--	--	0.20	3.54
	7/13	7.6	113	--	242	16	146	--	--	0.11	0.88
	7/15	7.8	111	--	235	17	152	--	--	0.03	0.58
	7/19	7.8	117	--	--	16	145	--	--	0.07	0.49
	9/16	7.10	78	7	210	48	157	15	3.9	0.50	1.1
	9/17	7.30	92	7	240	29	167	11	2.9	0.04	0.76
	9/18	7.30	97	--	240	26	166	10	2.7	0.35	0.95
	9/20	7.30	106	--	200	21	149	8.8	2.4	0.08	0.83
	9/22	7.55	110	7	240	24	163	7.4	2.1	0.05	0.77
	9/24	7.20	109	9	230	18	159	7.8	2.2	<0.02	0.65
	9/27	7.39	108	10	250	17	155	6.9	2.1	<0.02	0.77
	9/30	7.39	112	8	260	17	168	7.1	2.1	0.03	0.52
	10/4	7.42	119	7	260	18	169	5.5	1.9	0.02	0.92
Well P-4	7/19	8.0	110	--	--	39	182	--	--	0.02	0.10
	9/16	7.45	98	8	240	46	191	25	6.1	0.02	0.71
	9/17	7.80	105	4	270	36	183	23	5.8	<0.02	0.14
	9/18	7.80	108	--	260	40	184	24	5.8	<0.02	0.14
	9/20	7.80	109	--	230	38	171	25	5.6	<0.02	0.13
	9/22	7.65	109	3	280	39	178	24	6.1	0.04	0.14
	9/24	7.59	108	6	280	39	174	26	6.0	<0.02	0.13
	9/27	7.31	110	8	280	42	175	26	6.0	<0.02	0.12
	9/30	7.42	105	8	280	42	178	24	6.0	<0.02	0.39
	10/4	7.38	107	8	280	44	169	26	6.1	<0.02	0.12

[a]Grab samples used in establishing parameters.

TABLE 7. PILOT PROGRAM: WATER QUALITY DATA - RECEIVING STREAM[a]

Location	Date	pH	Alkalinity mg/ℓ $CaCO_3$	Acidity mg/ℓ $CaCO_3$	Specific Conductance μS/cm @25°C	Sulfate mg/ℓ	Total Dissolved Solids mg/ℓ	Calcium mg/ℓ	Magnesium mg/ℓ	Ferrous Iron mg/ℓ	Total Iron mg/ℓ
Laurel Lick Run Station A	9/17	7.35	25	3	270	95	206	21	7.3	<0.02	0.42
	9/20	7.00	16	-	150	44	119	16	4.4	<0.02	0.63
	9/22	6.90	16	3	180	53	127	16	5.2	<0.02	0.44
	9/24	6.85	16	4	170	48	147	15	5.3	0.04	0.42
	9/27	6.51	11	4	120	22	86	10	3.7	0.02	0.35
	9/30	6.57	12	4	140	34	105	12	4.2	<0.02	0.32
	10/4	6.70	12	4	130	35	94	12	4.2	0.04	0.34
Laurel Lick Run Station B	9/17	7.30	26	4	270	88	214	22	6.9	<0.02	0.34
	9/20	6.90	16	-	140	43	135	14	4.1	<0.02	0.70
	9/22	6.90	16	3	180	51	132	16	5.1	<0.02	0.40
	9/24	6.74	18	4	170	46	120	14	4.9	0.04	0.31
	9/27	6.40	11	4	110	23	84	10	3.6	<0.02	0.38
	9/30	6.62	12	4	140	33	96	11	4.2	<0.02	0.25
	10/4	6.57	12	3	130	34	90	11	2.9	0.03	0.21

[a]Grab samples used in establishing parameters.

TABLE 8. PILOT PROGRAM: WATER QUALITY DATA - MINE WATERS

Location	Date	Sample Type	pH	Alkalinity mg/ℓ $CaCO_3$	Acidity mg/ℓ $CaCO_3$	Specific Conductance uS/cm @25°C	Sulfate mg/ℓ	Total Dissolved Solids mg/ℓ	Calcium mg/ℓ	Magnesium mg/ℓ	Ferrous Iron mg/ℓ	Total Iron mg/ℓ
Main G Study Area Roof Water	7/12	Grab	7.9	124	--	268	18	178	--	--	0.02	0.02
	7/13	Grab	7.7	116	--	272	17	174	--	--	0.03	0.05
	7/15	Grab	8.1	120	--	255	18	173	--	--	0.05	0.06
	7/19	Grab	8.1	129	--	--	19	--	--	--	0.02	0.02
Underground L-2 Station	7/12	Grab	8.1	120	--	292	31	198	--	--	0.05	0.14
	7/13	Comp.	7.6	115	--	300	38	214	--	--	<0.02	0.69
	7/15	Comp.	7.9	119	--	310	27	223	--	--	<0.02	1.27
	7/19	Comp.	7.4	89	--	--	17	206	--	--	0.06	0.79
	9/15	Grab	7.50	113	--	240	34	177	20	4.1	0.02	0.36
	9/16	Comp.	7.75	114	--	250	36	188	17	4.1	0.02	0.23
	9/17	Comp.	7.85	116	4	260	30	190	18	3.9	<0.02	0.63
	9/18	Comp.	7.80	119	--	290	31	191	17	3.8	0.03	0.12
	9/19	Comp.	7.80	104	--	290	--	--	--	--	--	--
	9/20	Comp.	7.80	108	--	260	29	194	17	4.5	<0.02	0.18
	9/22	Comp.	7.85	119	5	260	32	193	15	3.7	0.05	0.34
	9/24	Comp.	7.82	116	5	270	28	186	25	3.8	0.05	0.57
	9/27	Comp.	7.82	116	5	260	28	179	20	3.9	0.04	0.47
	9/30	Comp.	7.70	116	5	270	32	178	20	4.0	0.04	0.60
	10/4	Grab	7.78	108	6	270	34	183	17	3.8	<0.02	0.21
	10/8	Grab	7.81	116	5	270	31	180	27	4.1	0.02	0.13
	10/10	Grab	7.82	113	9	260	33	175	19	4.1	<0.02	0.29
Treatment Plant Influent Line F-14	7/13	Grab	6.8	82	--	1300	636	1010	--	--	3.33	14.8
	7/19	Grab	7.6	100	--	--	645	1060	--	--	1.52	9.80
	9/17	Grab	7.00	85	7	1000	500	933	65	20	0.94	17
	9/20	Grab	6.80	102	--	1200	580	1030	66	20	<0.02	6.9
	9/22	Grab	6.80	68	33	1000	630	986	80	24	6.8	18
	9/24	Grab	6.85	92	35	1000	570	967	65	20	0.63	10
	9/27	Grab	6.71	106	27	1200	610	1060	62	18	0.67	3.9
	9/30	Grab	6.61	76	29	1100	600	990	75	22	1.7	18
	10/4	Grab	6.75	140	23	1200	626	1080	35	15	0.10	2.7
	10/8	Grab	6.69	72	27	1000	530	931	71	21	2.0	17
	10/10	Grab	6.79	83	22	990	490	876	42	20	3.4	12

SECTION 8

MINE DEWATERING SYSTEMS AND ALTERNATIVES TO DEWATERING

There are probably as many potential dewatering systems as there are underground coal mines, because the conditions that determine the best system are to a large degree site specific. However, there are two basic types of dewatering systems: vertical (conventional) wells drilled from the surface, and angled drain holes drilled from the underground mine openings. These dewatering systems are discussed in the following sections; in addition, some alternatives to dewatering and to current mine drainage and water treatment systems are suggested.

WELLS CONSTRUCTED FROM THE SURFACE

Vertical dewatering wells constructed from the surface either can be drained by gravity into the mine openings or they can be pumped individually to the surface. Parizek and others[11] present schematic drawings of these types of dewatering wells. Both systems can be effective in mine dewatering provided that fracture zones and/or aquifers overlying and adjacent to the mine can be tapped.

The gravity-drained wells are probably most effective where a relatively isotropic and more-or-less horizontal aquifer overlies the mine. Examples would be a sandstone bed or channel, or a limestone bed, with considerable areal extent, which both stores groundwater and transmits it to the mine. Under these conditions, groundwater response to well drawdowns is reasonably predictable and a regular well spacing can be used. The primary advantage of this system is that individual well pumps are not required, therefore capital and maintenance costs are restricted to those required for the pumps which raise the collected water to the surface. In the case of drift mines (mined up-dip), the system could be completely gravity drained, and therefore still more economical. Another advantage is the relatively small effect that drainage wells have on the ground surface. Disturbance at the surface is necessary only when wells are being constructed, and the only permanent surface facilities would be at the mine water discharge point(s). The discharge points could be located at the mine portal if all the drainage water is collected underground; if, instead, water is pumped to the surface, a single discharge point (a pumped well) could serve a group of drainage wells.

The primary disadvantage of wells drained by gravity into the mine is lack of flexibility in location. Since they have to be drilled into mine openings, they cannot be used to dewater in advance of mining. Also, where fracture flow dominates mine inflows, it may be impossible to intercept inflows from outside of or beyond the edge of mine openings, since fractures

above the mine may support only unsaturated flow, which cannot be intercepted by wells. Further, in longwall mining, the panels are so large that it may not be possible to penetrate important fracture zones to any significant degree--especially fracture zone intersections, which may be particularly important to dewater. Finally, penetrating mine openings presents some safety hazard because the breakthough could cause some amount of roof fall and there will be uncontrolled water inflows (along with methane gas, if present) until the drainage well is cased and hooked up to a header pipe. This may affect the mine ventilation system and will cause some disruption of normal mining activities.

In contrast, pumped wells can be drilled through pillars to some depth below the mine floors, or terminated at a safe distance above the mine roof; they can be drilled in advance of mining and/or located around the periphery of the mine. Aquifers can be tapped both above and below the mine, and there is more flexibility in locating pumped wells where they can tap fracture zones. The primary disadvantage, besides cost, would be that the facilities needed on the surface are relatively hard to protect against weather and vandalism. These facilities include powerlines to pumps and well discharge pipes. However, to balance this disadvantage, the well discharges can be collected and used on the surface for rural, municipal, or farm usage to replace or supplement water supply from existing wells and springs whose flow may have been affected by mining operations.

WELLS CONSTRUCTED UNDERGROUND

Standard mine air percussion drills can be used underground to drill effective drainage holes. These holes, about 6.4 cm (2.5 in.) in diameter, could be drilled vertically into the roof or into the floor where mine inflows are greatest, or they could be angled into the roof to intersect the the maximum density of fractures. Roof holes will drain by gravity, whereas floor holes would be pumped and would act in similar fashion to a construction well point system. The floor drains would be connected to a header pipe, with one pump serving a series of wells to control water inflow through the floor of the mine.

The roof holes would drain by gravity into a header pipe, which would transfer water to a central pumping station where it is pumped to the surface for discharge. These drainage holes can be designed either on a regular spacing and pattern or concentrated where inflows are encountered when driving entries and headings. Where fracture-dominated flow occurs, the drain holes should be oriented to intersect a maximum number of fractures. These holes can be drilled to arrays from both entries alongside longwall panels, and angled toward the panel and a low-angle drain just above the break line. Along the periphery of the mine, similar arrays of holes can be used to intercept inflows from the sides.

For maximum efficiency, a specially equipped crew of miners probably should be organized to drill drain holes. The drill should be mounted on mobile equipment using conventional drills with 1.2 m (4 ft) rods, carbide

bits, and standard 1.19 m^3/s (400 cfm) compressed air at 0.69×10^6 Pa (100 psi). To test the effectiveness of this system, trials can be made using equipment normally available in the mine for drilling rock-bolt holes.

A disadvantage of wells drilled from the mine openings is that as water inflows are encountered, they have to be handled individually in the conventional manner, at least temporarily. Treatment, however, may not be required if collection sumps and pumping stations for discharge to the surface are constructed prior to mine advance. These pumping stations would have to be over-designed to accommodate unanticipated inflows. Another disadvantage of this system is that while dewatering can be accomplished in advance of mining the longwall panels, it cannot be done in advance of driving headings and entries. These drains may produce more methane than normal, which may affect the ventilation system; however, this problem would be solved by proper collection of the gas at the drain head, and the methane removal system might serve wider applications than simply clearing the drains themselves.

There are two main technical advantages to this method of dewatering. The first is its flexibility: a system can be designed at the start of mining, based on exploration data, and easily adjusted as conditions dictate; or drainage holes can be constructed on a trial-and-error basis as mining advances. The second advantage is that the depth of the mine has little influence on the design of the system. The only adverse effect of depth on costs would be the increased size of pumps, and thus the amount of energy, required to pump to the surface. However, the depth of the seam similarly affects the costs of any of the dewatering systems. Thus, the merits of this approach outweigh the disadvantages.

DESIGN APPROACH

The design of a dewatering system can only be as good as the concept on which it is based. It is only as effective as the exploration, testing, and interpretation of the data upon which the design is based. For a new mine or extension of an existing mine, exploration of groundwater conditions should be combined with coal exploration. In particular, the exploration holes for coal can be designed for use in hydraulic testing and water quality sampling and preserved for use in monitoring or actually incorporated into the well dewatering system.

The surface geology of the area should be mapped with special emphasis on linear features and joint patterns. Both high-level and low-level aerial photography should be used. Existing wells and springs should be surveyed for yield, water quality, and potentiometric levels. Surface geophysical surveys should be performed to further delineate fracture zones and flow patterns. As mentioned previously in Section 6, surface resistivity combined with surface potential streaming surveys may be the best geophysical techniques to consider.

Exploration drilling should be programmed to prove the existence, extent, and system of fracture zones interpreted from surface surveys, as well as to explore the coal seam. Air-rotary drilling permits a rough estimate to be made of the water-bearing properties of rocks encountered if the holes are logged carefully and if the flow air-lifted from the holes is monitored as

drilling progresses. Exploration holes should be cased sufficiently so they can be used for water-level monitoring, and selected holes can be pump tested for hydraulic properties and water quality.

With these data, potentiometric surfaces can be mapped, which, when compared with fracture zones and pump tests, should indicate the areas of high potential inflows to the mine. The interpretation can be enhanced by mathematical modeling if the hydraulic data are reliable enough. A reasonable basis should then be available for designing a dewatering system and its technical and economic feasibility can be determined by comparing costs with benefits.

ALTERNATIVES TO MINE DEWATERING

After the completion of this study, it became obvious that development of AMD can be controlled or reduced by modifying existing mine drainage systems. In an existing mine, before any new type of dewatering program or mine drainage treatment program is established, a total water quality survey of mine inflows should be performed. Further, a monitoring net should be established for key parameters such as flow rate, pH, acidity, alkalinity, total iron, and specific conductance, and monitoring should be performed on a continuing basis until a basis for design is established. The monitoring program should then be converted as necessary as part of the control program. These surveys will establish the quantity and quality of the various underground flows and will show which flows do or do not meet direct discharge standards. This monitoring will indicate where water becomes degraded, if the quality is good to begin with, or where inflows occur that require treatment. Unfortunately, it is the experience of this study that monitoring of this type in most mines is non-existent. However, it is exactly the type of information that is needed before engineering decisions can be made in regard to the most effective solutions for water quality control.

With the above monitoring data, mine waters of different quality can be separated and handled differently. Good quality water can be intercepted and brought directly to the surface through bore holes and discharged without treatment or piped through the mine to the shaft or slope. The water quality information collected for the Lancashire No. 20 mine suggests that at least 60 percent of the total mine drainage could be intercepted within the mine and discharged without treatment. In contrast, it is now transferred from sump to sump where it degrades and is mixed with poor quality water from old workings, so that all of the mine drainage requires treatment. For example, it would have been a simple matter to: pump from the L-2 sump and G-11 sump into the L-4 pond; drill a hole from the surface into the L-4 pond and install a pump; then pump all of the Main G study area inflow to the surface and discharge it into Laurel Lick Run.

Often, the main source of poor quality water is from old or abandoned workings which may be difficult or too dangerous to enter. However, water from these workings can often be isolated and handled separately so that overall treatment costs are reduced even if the contributions from the active portion of the mine are low in iron or acidity (or alkaline).

Therefore, the experience developed during this study indicates that additional research is needed to survey the opportunities for control of AMD by determining water quality distribution throughout active coal mines. This may result in more cost-effective measures, especially for existing mines, than dewatering systems.

SECTION 9

COST-EFFECTIVENESS OF DEWATERING

INTRODUCTION

Objectives

The previous sections have demonstrated that groundwater can be intercepted by pumped surface wells before it enters an active underground coal mine. It has also been shown that dewatering can result in smaller amounts of AMD requiring treatment, since the water discharged will normally be of the same quality as existing groundwater, unless a nearby contaminated source such as flooded abandoned workings is also intercepted. The objectives of this section are threefold: First, to establish cost comparisons between conventional methods of handling and treating mine water and mine dewatering by pumped surface wells. Second, to extend the well dewatering cost data to show the influence on costs of variations in such elements as mine depth and water quality, which differ from one mine to another and among different areas of the same mine. Thirdly, to establish approximate costs for methods of dewatering other than by pumped surface wells.

Study Conditions and Methods

In order to make the cost comparisons, two sets of basic cost data were required. One set, pertaining to dewatering by pumped wells, was derivable from the study project. To arrive at the second set of cost data, pertaining to current methods of handling and treating mine water, and to cast the data into a form that would allow valid comparisons with the well dewatering costs, some research and mathematical manipulation were required.

The coal mining industry is highly competitive, and coal mine owners are reluctant to allow operating and cost data to become public. Therefore there is no official (or unofficial) industry standard of costs for in-mine water handling or water treatment which might serve as a base of comparison with costs established for the well dewatering project.

In fact, it may well be highly impractical to try to establish standard figures for the industry since there is great variation in physical factors from one coal mine to the next--size of mine, depth, water quality, engineering methods, etc.--so that costs can fluctuate greatly as a result of necessary differences in operations. The actual range of operating costs for one element of the mine water disposal system, say of water treatment alone, can be great. Figure 33, derived from data collected by Skelly & Loy (1973)[12] shows a range of costs per cubic meter for water treatment ranging from about 1¢ to

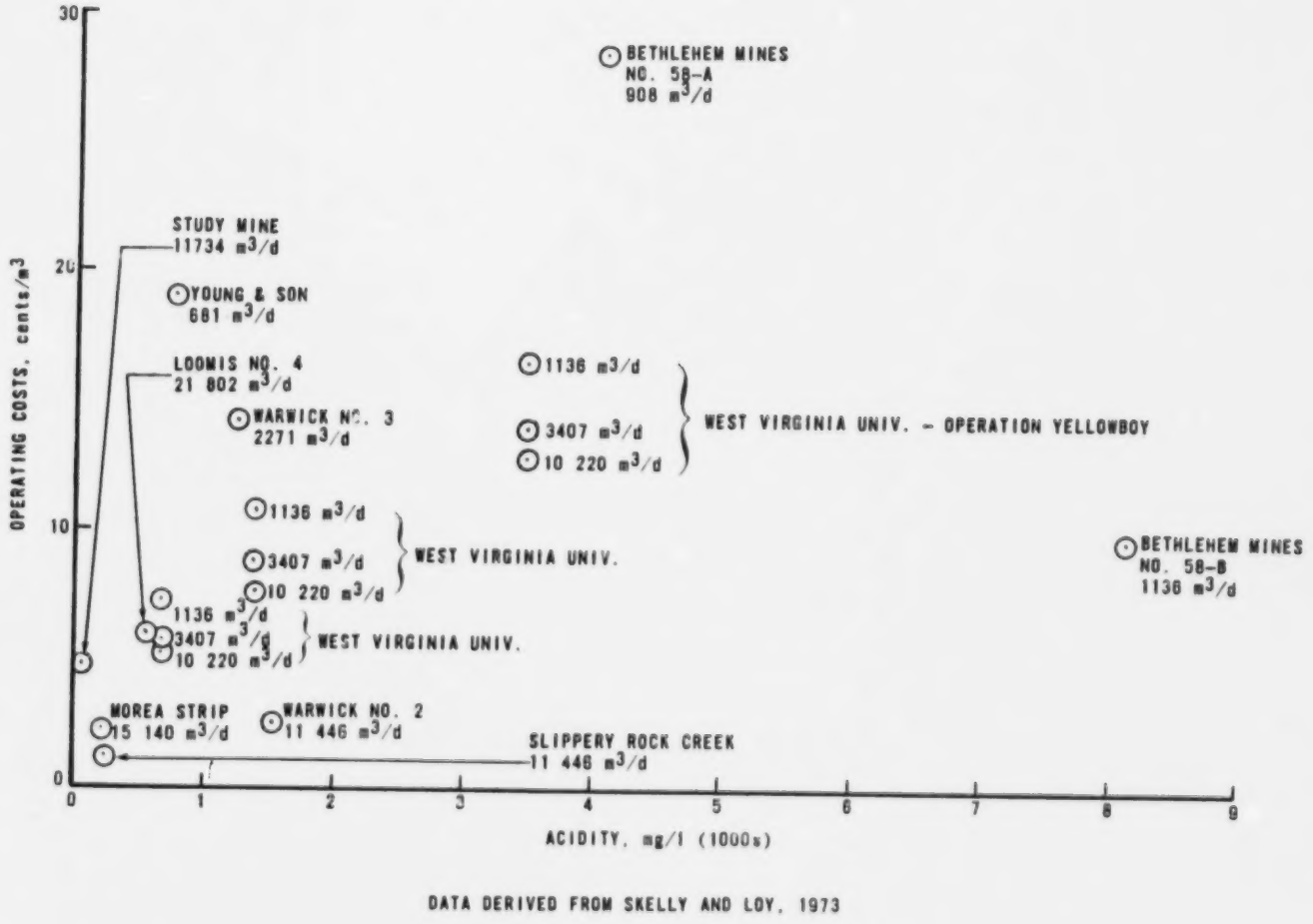


Figure 33. Operating costs for hydrated lime treatment facilities.

more than 20¢ depending on water quality. However, we do not know why the acidity of the mine water is high or low, how much it is affected by operations, and whether the higher acidities are necessary and unavoidable. That is to say, there are more physical variables influencing the final cost than simply the recorded acidity level of the water. Furthermore, differences in accounting practices can also contribute significantly to the range in costs for treating approximately the same quality and quantity of water. Accounting practices for all phases of mining industry activities are far from uniform, as is illustrated in a study presented at the Second Symposium on Coal Management Techniques and entitled "Survey of Accounting Principles, Procedures, and Practices in the Coal Industry."[13] The responses to the survey (Table 9) illustrate clearly the lack of uniformity in expensing and capitalizing standard costs, and the study as a whole demonstrates further variations in methods of depreciation, types of accounts, and the like.

TABLE 9. ACCOUNTING METHODS IN THE COAL INDUSTRY[a]

		During Production		
Item	Capitalized	Expensed	Mixed	Inventoried
Major repairs	2	9	9	
Standby equipment	17			1
Replacement parts	9	2		13
Slurry ponds (0-3 years of useful life)	7	4		
Construction changes	8	10		

[a]Source: Monterey Coal Co., 1976.

Because of these variations in physical conditions, lack of uniformity in accounting methods, and simple lack of published information, it is not possible to make valid comparisons of comprehensive and detailed costs between a single operation and the industry as a whole, or even between two individual operations. In evaluating the cost-effectiveness of well dewatering or of any other methods of water disposal, the primary comparison has to be among the methods available for the site in question. The considerations are both site-specific in the geotechnical aspect and system-specific in the accounting aspect. This is not to say that comparisons with data for other mines should be totally avoided, for useful comparisons can be made as long as their limitations are recognized.

The method of the present study, therefore, is to establish the costs of current water disposal methods at the study mine and then to draw limited comparisons between these data and costs for similar operations at other mines insofar as these comparisons are useful. The extent of 'usefulness' of this

data is its applicability to determining the cost-effectiveness of well dewatering under conditions other than those at the study mine.

COSTS OF MINE WATER DRAINAGE AND TREATMENT - STUDY MINE

Accounting Methods at Study Mine

To make the required cost comparisons feasible, the accounting data for mine water drainage and treatment at the Lancashire No. 20 mine must meet three conditions: (1) costs for the mine under study must be clearly identified and not grouped with other mines of the company; (2) capitalization and operating expenses must be clearly separated; and (3) the costs of water collection and transfer must be separate from costs for water treatment to improve quality. The Barnes and Tucker Company's accounting system meets these requirements, with a few exceptions which do not significantly affect the results of this study. Costs of mine water drainage and treatment are separated in the mine's accounting system by the method of cost accumulation, accounting, and payment. The two main categories of costs are capital investment and annual operating expenses. It is apparent that under capital investment all the costs for water treatment have been included (though these costs are not clearly segregated from one another), except for land acquisition, deferred investment, contingency, and fee. From the information available, it can be assumed that land acquisition costs were included under mine development and can be so attributed. Deferred investment refers primarily to the difference between original cost and the cost of replacing equipment and construction during the life cycle of the mine. These costs are therefore included in capital investment. Contingency and fee are not itemized and are introduced into this analysis as a factor of uncertainty. They generally run to 10 percent of the initial capital investment cost. Because the mine has been in operation for a number of years, the only capital costs for mine water drainage are accounted for as deferred investment in equipment and construction projects. The equipment is limited to that required for the removal of water from the mine.

The annual operating expense sections of the mine accounting system are not ideal in a few respects. The division into direct, indirect, and fixed expenses is not clear cut because the Company Service Department includes overhead in its charges to the mine operation for various services. The overhead apparently includes payroll overhead, union welfare, and general and administrative costs (G & A), the sum of which is about 90 percent. Taxes and insurance have not been noted; for this analysis they are set at 2 percent of the initial capital investment cost. Power and water costs are not tabulated separately for the mine, preparation plant, and water treatment plant, and this overall cost is in turn combined with similar costs for other mines as a single, tabulated value. For purposes of this analysis, these values were estimated. The methods of estimation are presented as part of the discussion of these costs below.

Labor cost is not assigned to each mine for the two personnel employed to oversee the operation and effectiveness of the water treatment of the four mines owned by the company in the same local geographic area. However, this

cost was assumed (probably with reasonable accuracy) to be the total yearly salary of the two, with the addition of overhead at 90 percent, divided by four.

An attempt was made to determine the completeness of cost entries, and to identify joint costs and hidden costs. In most cases, missing cost entries are recorded elsewhere in the accounting system as a joint cost. Joint costs and missing entries were calculated using standard techniques. There are certain hidden costs which result from entries being present but not complete, particularly in connection with some types of labor costs. For example, only pumper personnel charge time to the drainage account. In a wet mine, such as this one, heavy influxes of water occasionally occur which require the work of all the face section personnel for two or three days. The actual time costs for one single such event can be equivalent to 10 to 30 percent of the annual pumpers' labor cost. At least one such event occurred during the 1976 fiscal year (the period of the dewatering study). Another unrecorded labor cost is the use of injured and other unassigned personnel to help in repair of pumps on the surface. To compensate for these unrecorded costs, an error of plus 20 percent of pumper labor cost is added to taxes and insurance (2 percent of initial capital investment) as an upper range of operating expenses for drainage and water treatment.

Capital Investment and Annual Operating Costs

The water-related mine costs studied in this analysis are for the company's 1976 fiscal year from October 1975 through September 1976. The detailed costs, as provided by the company, are presented on Tables 10 through 13. Tables 10 and 11 itemize annual operating expenses for water collection (drainage) and water treatment, respectively. Table 12 shows special work orders for both drainage and water treatment over the 12 months of the company's fiscal year. Table 13 presents the items capitalized as depreciable assets for both drainage and water treatment.

Table 14 summarizes capital investment costs in which contingency and fee are added and total depreciation to date is noted. The total investment cost for drainage and water treatment is about $701,000, or $771,000 with contingency and fee. The investment cost for drainage is about $310,000, or $342,000 with contingency and fee. The investment cost for water treatment is about $391,000, or $430,000 with contingency and fee.

Table 15 is a short summary of annual operating expenses in which estimates for missing expense entries are added to the company's entries. The items of depreciation for drainage and water treatment are minor entries. Taxes, insurance, and unrecorded labor are calculated as operating costs which are not recorded by the mining company. The percentage increase represented by these costs is given. The unrecorded labor under drainage is estimated from activities throughout the year above and below ground, including one instance in which 2 sections of 12 men worked 3 days to control a particularly heavy flow. This single incident accounts for 10 percent of the entry. The accuracy of such entries can be questioned, yet an expense should be entered

TABLE 10. SUMMARY OF DRAINAGE ANNUAL OPERATING EXPENSES

	Account No.	Cost Source	Description	Actual Costs Recorded in Water Related Accts. Detail	Actual Costs Recorded in Water Related Accts. Total	Allocation of Costs Charged Elsewhere
Labor	315	P/R Journal	Labor Cost for UMWA employees classified as pumpers		$92,523	
Fringe Benefits	-	--	Estimated @ 90% of labor			$83,270
Material						
Direct Charges	316	Acct. Pay (AP) Journal	Outside charges for Material & Labor			
			Repairs to pumps	$36,740		
			Purchase of New Pumps	9,930		
			Valves	14,895		
			Couplings, motors & other parts	22,839		
		AP & Supplyhouse	Fiberglass pipe	14,895	99,299	
Work Orders		Std. Journal	Parts & Labor & Overhead chgd. from Company Service Dept.			
Blanket Order		#104	Routine Maint - Small Drainage Pump Motor	$15,088		
Blanket Order		#112	Routine Repair - #10 Labour pumps	-		
Blanket Order		#113	Routine Repair - #23 Labour pumps	1,478		
Blanket Order		#114	Routing Repair - All other small pumps	159		
Special Orders		Various	See Table 11	6,807	23,532	
			Total Material and Expense		122,831	
			Total Actual Cost Recorded - Underground Drainage		$215,354	

TABLE 11. SUMMARY OF WATER TREATMENT ANNUAL OPERATING EXPENSES

	Account No.	Cost Source	Description	Actual Costs Recorded in Water Related Accts. Detail	Actual Costs Recorded in Water Related Accts. Total	Allocation of Costs Charged Elsewhere
Labor		None	No specific employees directly assigned - plant operated by Service Department and Labor charged via Blanket Work Order #103, below			
Fringe Benefits	--	--	Charged via Work Order #103, see Table 11			
Material						
Direct Charges	394-4	Acct. Pay (AP) Journal	Outside charges for Material & Labor			
			Lime	$39,170		
			Pumps	7,500		
			Parts & Repairs to Pumps, etc.	8,325		
			Valves	1,665		
			Equipment Rental - Sludge Removal	20,000		
			Seeding & Mulching	6,684	$83,344	
Work Orders	--	Std. Journal	Parts, Labor & Overhead chgd. from Company Service Dept.			
Blanket Order		#103	Operation & Routine Maint. of facilities	$72,982		
Special Orders		Various		20,181	93,163	
			Total Actual Cost Recorded - Water Treatment		$176,507	

TABLE 12. WORK ORDERS - WATER PREATMENT AND DRAINAGE - FISCAL 1976

	Oct.	Nov.	Dec.	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Total
WATER TREATMENT A/C C 349-4													
[a]W/O #103 - Blanket Order	$3,356	$3,840	$3,902	$3,673	$3,352	$9,706	$4,962	$6,055	$9,251	$9,533	$6,877	$8,475	$72,982
#9747 - Construct pumping station to convey cleaning plant water to #2 Basin (incl. ditch, pipe, pump, foundation, etc.)		502	8,665	10,828	186								20,181
TOTAL	$3,356	$4,342	$12,567	$14,501	$3,538	$9,706	$4,962	$6,055	$9,251	$9,533	$6,877	$8,475	$93,163
DRAINAGE - A/C C 316													
W/O #9869 - Repair 25 hp pump starter			531										531
#9864 - Repair 40 hp electric motor			401										401
#9975 - Repair 300 hp Emco pump starters					515								515
#9976 - Mount motor & pump on base					744								744
#10114 - Repair 20 hp pump motor							733						733
#10134 - Repair 300 hp pump motor								257					257
#10135 - Repair 15 hp pump motor								643					643
#10184 - Repair Acme compressor									2,608				2,608
#10286 - Repair 50 hp trash pump motor											375		375
Sub Total			932		1,259		733	900	2,608		375		6,807
W/O #104 - Blanket Order	183	1,451	1,744	440	746	1,913	4,036	1,906	899	383	900	482	15,088
#113 - Blanket Order	716	762											1,478
#114 - Blanket Order							159						159
TOTAL	$ 899	$2,213	$2,676	$ 440	$2,005	$1,913	$4,928	$2,806	$3,507	$ 383	$1,275	$ 482	$23,532

[a]All work orders include labor, fringe, and all departmental overhead charged via departmental rate applied to actual hours.

TABLE 13. SUMMARY OF ITEMS CAPITALIZED AS DEPRECIABLE ASSETS - DRAINAGE AND WATER TREATMENT

	Depreciation Life	Purchase Date	Cost	Accumulated Depreciation	Net Book Value	Comments
UNDERGROUND DRAINAGE						
Pumps - Various, incl. Deep-Well Pumps	10 yr.	1964 to 1976	$88,075	$24,054		Cost of Equipment, Only
Pipe - Fiberglass & PVC	10 yr.	1964 to 1976	69,885	25,625		Cost of Equipment, Only
Motors, Starters, etc.	10 yr.	1964 to 1976	39,271	9,164		Cost of Equipment, Only
Various Underground Projects Such as: Sump areas, centrifugal pumps, bore holes, bulkhead of old mine section, etc. - incl. pipe pumps, etc.	20 yr.	1966 to 1968	113,249	40,867		Includes Vendor's cost for equipment, contractors and company labor and Service Dept. charges[a]
Total Underground Drainage			$310,480	$99,710	$210,770	
WATER TREATMENT FACILITIES						
Old Treatment Facility	20 yr.	1967	26,552	21,319		Outside contractor and company labor
New Treatment Facility	20 yr.	1971 to 1972	293,636	16,864		Includes cost of outside services for site preparation, engineering, construction and equipment plus charges for company Service Dept.
Hydraulic Rake System - Equipment	20 yr.	1975 to 1976	35,512	3,551		Cost of Equipment, Only
- Installation	20 yr.	1975 to 1976	35,052	1,753		Charges from company Service Dept.
Total Water Treatment Facility			$390,752	$43,487	$347,265	

[a]Charges from Company Service Dept. includes Labor, Fringe, and All Departmental Overhead.

TABLE 14. SHORT SUMMARY OF CAPITAL INVESTMENT

Function	Investment Cost[a]	Accumulated Depreciation	Depreciation Cycle	Contingency/ Fee
Drainage	$310,480	$99,710	10 years -pumps 20 years -projects	$31,048
Drainage with Contingency/Fee	341,528	_____		_____
Water Treatment	390,752	43,487	20 years	39,075
Water Treatment with Contingency/Fee	429,827	_____	_____	_____
Total Cost[b]	$701,232	_____	_____	_____

[a]Although data are presented to the nearest dollar, rounding to hundreds or thousands of dollars would be appropriate for subtotals and totals, as these numbers imply a precision not justified by the use of a single year's costs, or by the method of accounting, or by the use of estimates where data are missing.

[b]Total Cost including Contingency/Fee: $771,355.

TABLE 15. SHORT SUMMARY OF ANNUAL OPERATING EXPENSES

Type of Expense	Drainage Cost	Water Treatment Cost
Labor	$92,523	--[a]
Fringe Benefits	83,270	--[a]
Material	99,299	$83,344
Work Orders	23,532	93,163
Power and Water	196,430[b]	--[b]
Depreciation	10,000[c]	10,000[c]
Subtotal	$505,054	$186,507
Taxes/Insurance @ 2% Capital Invest.	6,830	8,597
Unrecorded Labor @ 20% Labor	18,505	14,250[d]
Subtotal	25,335[e]	22,847[f]
Total	$530,389	$209,354

[a]Assigned to 4 mines - costs not separated out.

[b]The figure entered under drainage costs may be taken as for power only. Costs of power for water treatment and of water for both drainage and water treatment are within the error of the calculation of drainage power cost. These estimates are not included in order to make the power estimate even more conservative.

[c]Depreciation is a straight line apportionment of accumulated depreciation back to initial date of installation of major investment for water treatment (1972) and back to initial introduction and recording of pumps (1964).

[d]($30,000 + $27,000) ÷ 4 mines.

[e]5.0% increase.

[f]12.2% increase.

to represent the fact that some charges accrued but are not recorded for the activity. From Table 15, the total annual operating cost of water collection and treatment is approximately $740,000.

The costs of electricity in operating the pumps are not separated from other electrical costs in the operation of the mine. These energy costs were estimated by obtaining the type and number of pumps with their full-load horsepower rating. Allowance was then made for daily usage. The horsepower was then converted to kilowatt hours and costs were established per kilowatt hour. Finally, an estimate was made of percentage of full load at which the pump is set. The resulting computed annual energy costs for the various pumps are shown on Table 16. Also as shown on Table 16, the total annual mine pumping cost is $157,158. The annual demand charge is not shown on Table 16, but was calculated to be $39,272 and the total annual cost therefore is $157,158 (from Table 16) plus $39,272, or $196,430. This total, shown on Table 15, is 27 percent of the total cost for electricity of the study mine. These figures confirm a gross estimate of 25 percent furnished by a knowledgeable employee of the mining company.

The estimated average annual discharge from the mine based on pump records is 13 400 m^3/d (3.54 mgd). This value is only a gross measure because it is based on a recording of the hours the pump was on and its rated operating displacement. The pump flow from one of the pumps (F-14) was measured with a flow meter over a period of 119 days, and the resultant average rate was 4400 m^3/d (1.15 mgd). Similarly, the average value estimated from records for this period was 5100 m^3/d (1.34 mgd). As a result, the estimate based on the F-14 pump records was about 14 percent too high. Therefore, the flow values based on pump records were reduced to 11 700 m^3/d (3.1 mgd), and this value was used in cost calculations.

An important calculation for comparison purposes is the operating cost of water collection (drainage) and water treatment. This cost is calculated as follows:

$$\Delta = \frac{100\ C\ (\text{annual})}{D \times P}$$

Where C is the annual operating cost
D is the number of operational days in the year
P is the daily pumping rate in m^3
Δ_D is the water collection costs in ¢/m^3
Δ_{WT} is the water treatment costs in ¢/m^3

$$\Delta_D = \frac{\$530{,}000 \times 100}{365 \times 11{,}730} = 12.4\ ¢/m^3, \text{ or } 46.8\ ¢/1{,}000\ \text{gal}$$

$$\Delta_{WT} = \frac{\$209{,}000 \times 100}{365 \times 11{,}730} = 4.9\ ¢/m^3, \text{ or } 18.5\ ¢/1{,}000\ \text{gal}$$

TABLE 16. ANNUAL MINE COST OF ELECTRICITY FOR UNDERGROUND DRAINAGE

Flow Point	Type of Pump	No. of Pumps	Power per Unit (hp)	Total[a] Load (hp)	Period of Full Load (h/d)	Energy Consumption kWh/d	Daily Cost[b]	Annual Cost[c]
F-14	1500 gpm Gould	1	300	255.0	24	4,565.5	$100.44	$36,660.60
F-1	1500 gpm Gould	1[c]	300	255.0	24	4,565.5	100.44	45,825.00
K-7	2500 gpm Gould	1	300	255.0	5	951.2	20.93	7,639.45
Slant	1000 gpm Wilson Snyder	1	125	106.3	24	1,903.2	41.87	15,282.55
Slant	2500 gpm Hazelton	1	100	85.0	24	1,521.8	33.48	12,220.20
Main-C	1000 gpm Flood City	1	50	42.5	2	63.4	1.39	507.35
G-5	1000 gpm Flood City	1	50	42.5	3	99.1	2.18	795.70
F-17	1000 gpm Flood City	1	50	42.5	8	253.6	5.58	2,036.70
G-7	1000 gpm Hydramatic	1	50	42.5	12	380.5	8.37	3,055.05
G-9	1000 gpm Hydramatic	1[c]	50	42.5	12	380.5	8.37	3,055.05
K-12	1000 gpm Hydramatic	1	50	42.5	12	380.5	8.37	3,055.05
Fresh H_2O	1000 gpm Gould	1	50	42.5	12	380.5	8.37	3,055.05
G-10	150 gpm 23 Labour	1	15	12.8	24	229.2	5.04	1,839.60
G-11	150 gpm 23 Labour	1	15	12.8	12	114.6	2.52	919.80
F-13	150 gpm 23 Labour	1	15	12.8	24	229.2	5.04	1,839,60
F-17	150 gpm 23 Labour	1	15	12.8	16	152.8	3.36	1,226.40
F-20	150 gpm 23 Labour	2	15	25.5	24	456.6	10.05	3,668.25
K-7 Belt	150 gpm 23 Labour	1	15	12.8	24	229.2	5.04	1,839.60
Main H	50 gpm 10 Labour	1	5	4.3	8	25.7	0.57	208.05
F-20	50 gpm 10 Labour	1	5	4.3	8	25.7	0.57	208.05
Faces	50 gpm 10 Labour	40	5	170.0	12	1,521.8	33.48	12,220.20
	Totals	64[c]	2140[c]			18,430.1	$405.46	$157,158.10

[a] Full load = 0.85 is included in this calculation.

[b] Total rate per kWh is $0.007 basic plus $0.015 fuel adjusted, for a total of $0.022.

[c] There is a second pump on standby. Figures for occasional use of this pump are included in total.

A summary of totals of annual operating expenses and capital investment for fiscal 1976 is given in Table 17. The water treatment required greater capital investment and underground drainage required greater annual expense.

TABLE 17. MINE COST SUMMARY

Type of Cost	Water Collection	Water Treatment	Total
Annual Operating Expenses	$530,000	$209,000	$739,000
Capital Investment	$342,000	$430,000	$772,000

Major Cost Contributors

In this section, the major contributors to the costs of the study mine's current water disposal system are presented, as far as they can be discriminated. Contributors to drainage costs are discussed first, and contributors to treatment costs second. The separation between operating expenses and capital investment is maintained in each case.

The cost contributors in annual operating expenses for underground drainage are presented in Table 18. There are two functional categories of underground drainage: (1) collection and transfer of water to the bore hole, and (2) pumping to the surface. A key separation is made between these two sets of cost contributors to permit comparison to the dewatering major cost contributors presented later. Although costs are not recorded according to this categorization, approximate estimates can be made. The procedure is to calculate the percentage of each cost contributor (e.g. power,) that is used either to collect and transfer water or to pump water to the surface.

TABLE 18. UNDERGROUND DRAINAGE ANNUAL OPERATING EXPENSES - COST CONTRIBUTORS

Item	Power	Labor	Material	Orders	Deprec. and Taxes	Total
Water Collection and Transfer	$92,702	184,583	99,299	23,532	8,415	$408,531
Pumping to Surface	$103,728	9,715	0	0	8,415	$121,858
Total	$196,430	194,298	99,299	23,532	16,830	$530,389[a]
% of Total	37	36	18.7	4.4	3.2	100%

[a]See Table 15.

Power costs are the main contributor (37 percent) to the annual operating expenses shown on Table 18. In calculating power costs, Table 16 is key because it presents all the pumps with their purpose noted. Three pumps (F-1, F-14, Main C) are used to discharge water from the mine to the surface. Using the power costs of these three pumps on Table 16, the cost of electricity for pumping to the surface is computed to be 52.8 percent of the total cost of electricity on that table:

$$\frac{\$36,660 + \$45,825 + \$507}{\$157,158} = \frac{\$82,992}{\$157,158} = 52.8\%$$

Using this percentage to prorate the annual demand charge and adding the result to the energy charge, the total annual power cost to pump the water to the surface is:

$$\$82,992 + (0.528 \times \$39,272) = \$103,728 \text{ (as shown on Table 18)}$$

The percentage of the energy charge for collection and transfer is 47.2 percent (100 percent less 52.8 percent) and thus the total power cost for collection and transfer is:

$$(\$157,158 - \$82,992) + (0.472 \times \$39,272) = \$74,166 + \$18,536 = \$92,702$$
(as shown on Table 18)

The next major contributor of underground drainage annual expenses is labor (36 percent). In this case, however, the operation and maintenance of the water collection and transfer system is far more extensive than simply the maintenance of three mine discharge pumps. The number of pumps is greater and the distance traveled for maintaining the operation is far greater. It is estimated that the labor cost to pump water to the surface is about 5 percent of the total labor cost for underground drainage or $9,715, leaving $184,583 to cover labor costs for collection and transfer of water.

The next largest cost contributor is material (18.7 percent). All of the cost is allocated to water collection and transfer because the pumps and piping are capitalized. Thus, material is set at 100 percent water collection and transfer or $99,299. The smaller cost contributors are work orders (4.4 percent), and depreciation and taxes (3.2 percent). Work orders relate totally to water collection and transfer costs, while depreciation and taxes are allocated at 50 percent to each of the two functions.

Of the two sets of functional cost contributors in total underground drainage annual expenses (Table 18), water collection and transfer is by far the larger, representing 77 percent ($408,531) of the total cost, while pumping to the surface costs only 23 percent of the total ($121,858).

The underground drainage capital investment costs (Table 17) could not be separated into these two functional cost contributors. The cost contributors to drainage capitalization are primarily in equipment and piping. Equipment accounts for at least two-thirds of the cost. Drilling of boreholes, construction of bulkheads, and construction of sump areas are the source of the remaining one-third. Labor is included in the construction projects only.

The annual operating expenses of water treatment are separable by the usual divisions of labor, depreciation, repair, etc. The expense tabulations provided by the mine company are presented in Table 19. The major cost contributor in this table is the item "labor, material, and repair" because it cannot be broken down any further. "Repair" includes maintenance and operation as well as actual repair. "Lime" is separated from "material" and is the next largest cost contributor. The third largest is "sludge disposal". This entry includes only the truck rental, and not labor or any other source of expense. "Miscellaneous" includes emission tests and taxes (Table 15).

TABLE 19. WATER TREATMENT ANNUAL OPERATING EXPENSES - COST CONTRIBUTORS (1976 FISCAL YEAR)

	Labor, Material, and Repair[a]	Lime[b]	Deprec.[c]	Sludge Disposal[b]	Misc.[a]	Total[c]
Cost ($)	124,923	39,170	10,000	20,000	15,261	$209,354
% Total	59.7	18.7	4.8	9.6	7.3	100.1%[d]

[a]Derived from information shown on Tables 11 and 15.

[b]See Table 11.

[c]See Table 15.

[d]Rounding error.

The capital costs for water treatment could not be separated into functional cost contributors. The various parts of the treatment plant such as thickeners, sludge ponds, etc., could not be identified from the cost data provided by the mining company.

Summary

In the preceding discussion, water-handling costs at the study mine have been analyzed to identify cost factors that will permit comparison with other systems in the industry and with the pilot dewatering operation. The analysis is summarized in chart form on Figure 34. The water-handling costs have been divided first into the conventional categories of capital investment and annual operating expenses. Each of these categories is then subdivided into costs for underground drainage (collection and transport of water underground) and for water treatment. This second division is key to the later comparisons, since the water treatment costs become a means of comparing the study mine to the industry, and also constitute the primary savings to be anticipated in a well dewatering system. Subsequently, cost contributors to annual operating expenses for both underground drainage and water treatment have been identified or (where necessary and possible) estimated, again to facilitate later

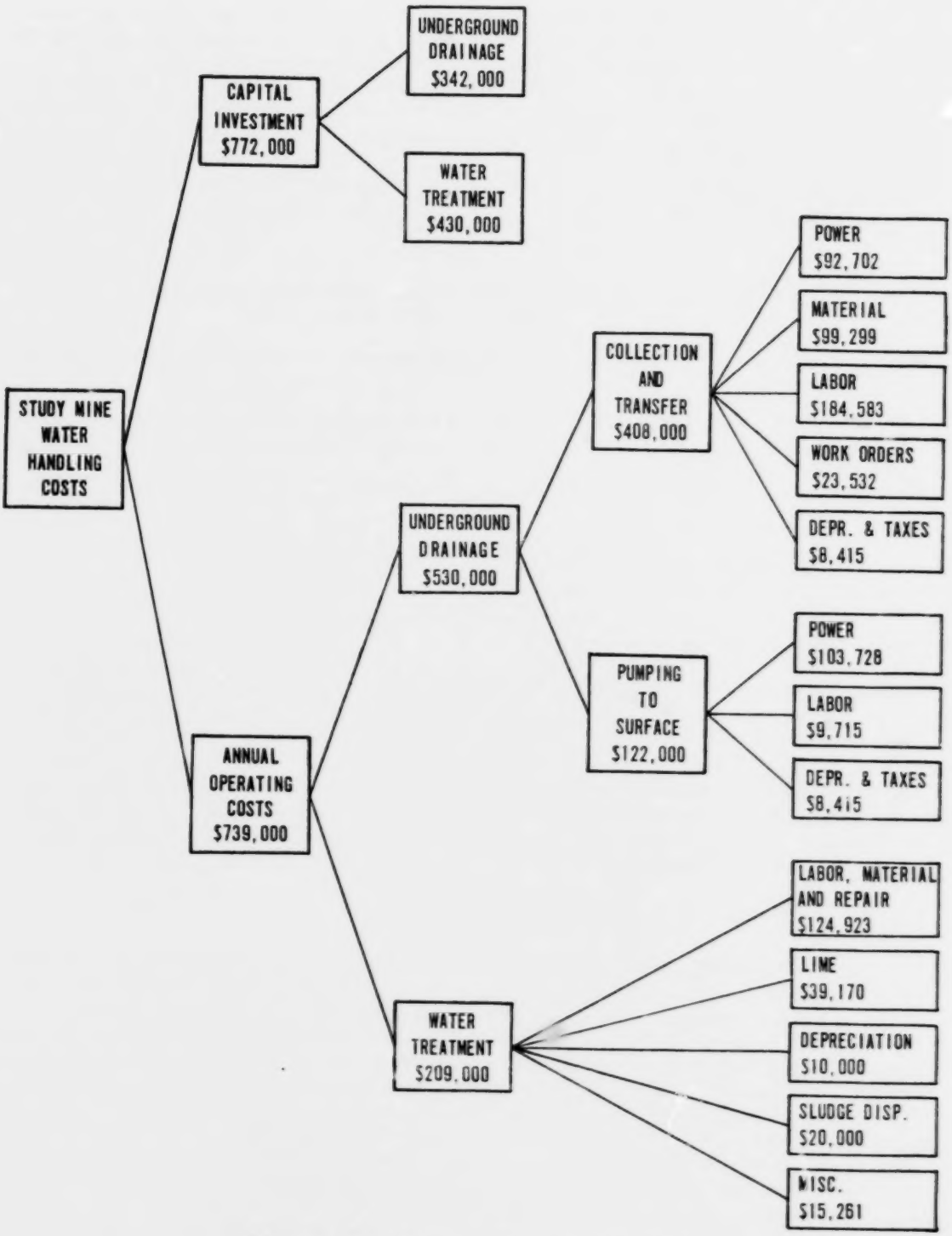


Figure 34. Summary of cost analysis.

comparisons. The available data on capital investment were not appropriate for identification of cost contributors to capitalization of either underground drainage or water treatment.

INDUSTRY COSTS

As noted earlier in this section, data on costs in the coal mining industry are infrequent. There appear to be no data published on costs of underground drainage, and therefore we have to assume that costs at the study mine are somewhat representative for the industry.

On the other hand, the question of treatment of AMD has been a focus of public attention as well as a concern of the industry for some 10 to 15 years, and reliable cost data have emerged, especially from the Operation Yellowboy Program, reported by Charmbury et al., 1967[14]. In this program conducted during the 1960's at the University of Virginia Mine No. 1, a lime treatment facility was operated under study conditions, and consistent accounting methods were used to determine capital and operating costs for various water quality conditions and plant capacities. These cost data have been used by Holland et al.[15] with reference to treatment plant design, and Holland's work has been used by Shumate et al.[16] to develop predictive models for hydrated lime neutralization capital investment and annual operating costs. Cyrus Wm. Rice & Company[17] also use these data in evaluating the economics of lime neutralization as a method of water treatment. Doyle, Bhatt, and Rapp[18] draw on all of these earlier works (except Shumate) in their review of water treatment methods, and they present several tables and charts which provide a useful industry context for the study mine. All cost elements in Doyle et al. are stated in 1972 dollars. Since study mine costs are based on 1976 accounts, study mine costs have been converted to 1972 dollars in the following comparative discussions using the construction contract cost index presented below.

Construction Contract Cost Index from 1967-1976

Year	Index
1967	100
1968	113.2
1969	123.7
1970	123.1
1971	145.4
1972	165.3
1973	179.5
1974	169.7
1975	166.0
1976	168.1

Tables 20 and 21 together present a broad view of treatment costs in the industry. They show capital and operating costs, respectively, for ten treatment operations and also for the Operation Yellowboy study plant. Table 22, based on Operation Yellowboy data, shows a range of cost contributors to costs of hydrated lime treatment at different levels of acidity and plant capacity. Figure 35 and 36, based on the same reference data, plot capital costs and

TABLE 20. SUMMARY OF CAPITAL COSTS - CONVENTIONAL LIME NEUTRALIZATION PROCESS[a]

Mine	Design Flow Rate m^3/Day	Total Acidity mg/ℓ	Capital Costs	
			Total Cost	$/$m^3$
Bethlehem Mines Co. No. 58-A	908	4 080	$ 347,200	$382.38
Bethlehem Mines Co. No. 58-B	1 136	8 150	423,300	372.53
Young & Son	681	770	229,900	337.59
Morea Strip	15 140	190	657,400	43.42
Blue Coal Corp. Loomis No. 4	21 802	560	1,094,000	50.18
Duquesne Light Co. Warwick No. 3	2 271	1 250	229,700	101.14
West Virginia University School of Mines Mine No. 1 (Operation Yellowboy)	1 136	3 500	---	---
	1 136	1 400	---	---
	1 136	650	---	---
	3 407	3 500	---	---
	3 407	1 400	---	---
	3 407	650	---	---
	10 220	3 500	---	---
	10 220	1 400	---	---
	10 220	650	---	---
Duquesne Light Co. Warwick No. 2	11 446	1 560	582,000	50.85
Commonwealth of Pa. Slippery Rock Creek Treatment Plant	11 446	240	750,000	65.53
Rausch Creek Mine Drainage Plant	37 850	---	1,747,380	46.17
Mountaineer Coal Co.	2 725	250	120,000	44.04

[a]Source: Skelly and Loy, 1973.

TABLE 21. SUMMARY OF OPERATING COSTS - CONVENTIONAL LIME NEUTRALIZATION PROCESS[a]

Mine	Design Flow Rate m^3/Day	Total Acidity mg/ℓ	Operating Costs	
			Annual Cost	¢/1000m^3 per mg/ℓ acidity
Bethlehem Mines Co. No. 58-A	908	4 080	$ 95,250	7.0
Bethlehem Mines Co. No. 58-B	1 136	8 150	140,000	4.1
Young & Son	681	770	47,400	24.8
Morea Strip	15 140	190	126,571	12.1
Blue Coal Corp. Loomis No. 4	21 802	560	475,000	10.7
Duquesne Light Co. Warwick No. 3	2 271	1 250	117,500	11.3
West Virginia University School of Mines Mine No. 1 (Operation Yellowboy)	1 136	3 500	68,448	4.7
	1 136	1 400	44,236	7.6
	1 136	650	30,223	11.2
	3 407	3 500	172,463	4.0
	3 407	1 400	108,405	6.2
	3 407	650	73,913	9.1
	10 220	3 500	477,968	3.7
	10 220	1 400	290,723	5.6
	10 220	650	196,607	8.1
Duquesne Light Co. Warwick No. 2	11 446	1 560	109,715	3.2
Commonwealth of Pa. Slippery Rock Creek Treatment Plant	11 446	240	51,000	5.1
Rausch Creek Mine Drainage Plant	37 850	---	---	---
Mountaineer Coal Co.	2 725	250	---	---

[a]Source: Skelly and Loy, 1973.

TABLE 22. COMPARISON OF COST CONTRIBUTORS: STUDY TREATMENT FACILITY VS. OPERATION YELLOWBOY RESULTS

Study Treatment Facility[a]	Labor, Material and Repair	Lime	Depreciation	Sludge Disposal	Misc.	Total
100 mg/ℓ acidity - 3.7 mgd capacity						
% of Annual Operating Expenses	59.7	18.7	4.8	9.6	7.3	100.1[c]

Operation Yellowboy Facility[b]	Labor	Repair	Lime	Plant Cost	Sludge Disposal	Misc.	Total
150 mg/ℓ acidity - 2.7 mgd capacity	15.3	4.3	19.7	43.6	4.3	12.3	100
	19.6						

[a]From Table 19.

[b]Developed from costs data extrapolated from Table 20.

[c]Rounding error.

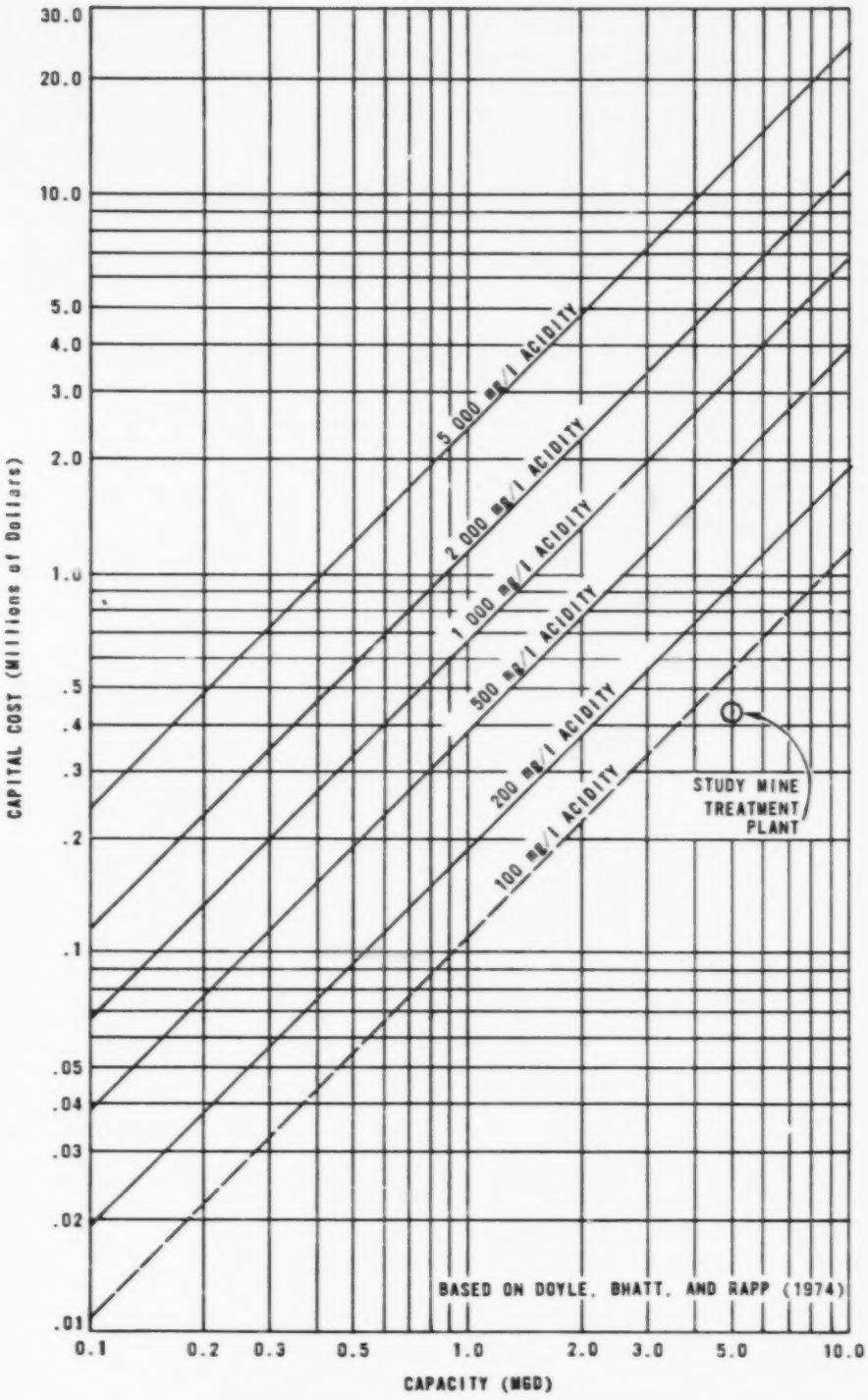
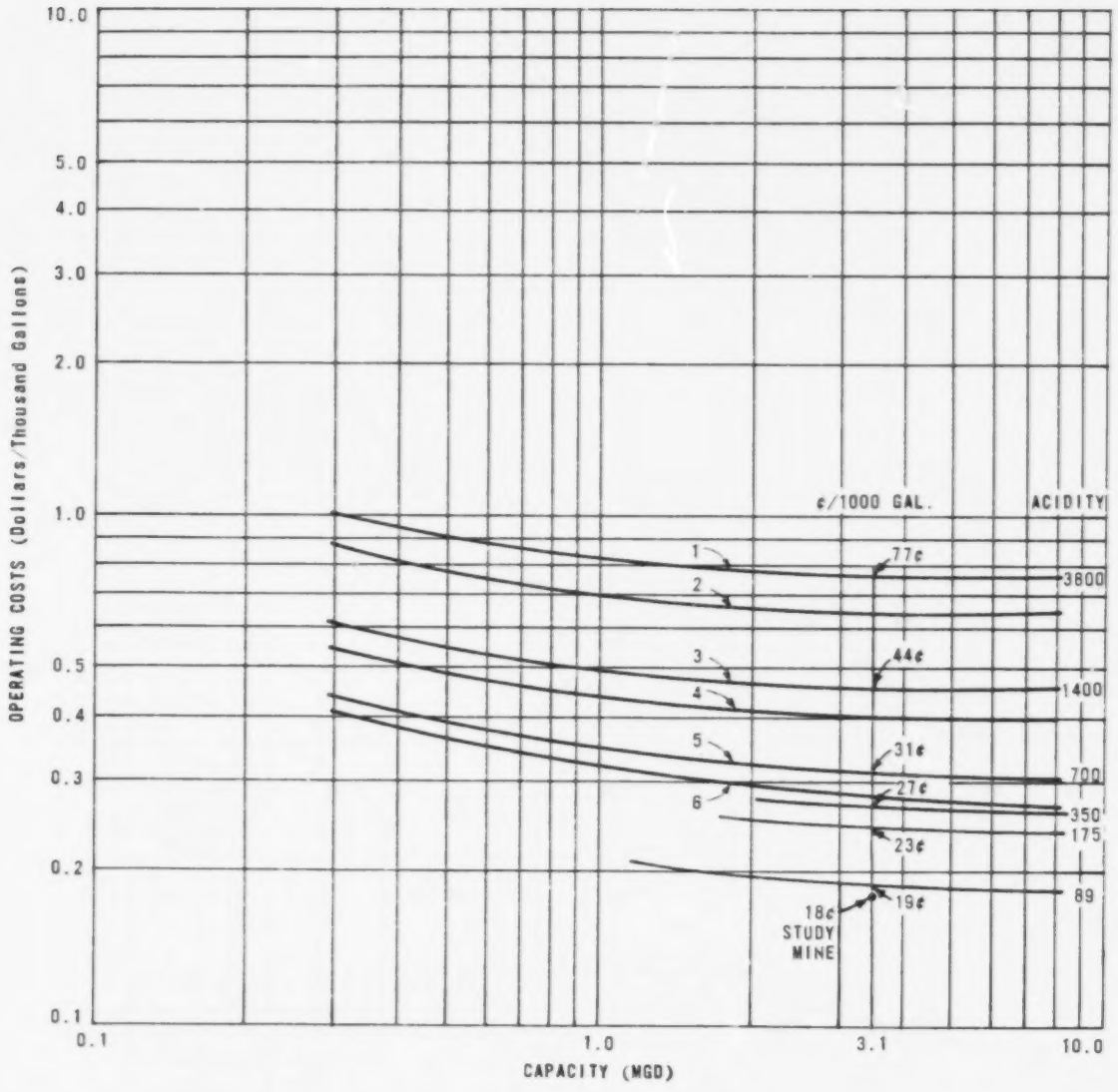


Figure 35. Capital cost of treatment plant vs. plant capacity - hydrated lime treatment with sludge disposal.



1-HIGHLY ACIDIC (WITH SLUDGE DISPOSAL)
2-HIGHLY ACIDIC (WITHOUT SLUDGE DISPOSAL)
3-MODERATELY ACIDIC (WITH SLUDGE DISPOSAL)
4-MODERATELY ACIDIC (WITHOUT SLUDGE DISPOSAL)
5-WEAKLY ACIDIC (WITH SLUDGE DISPOSAL)
6-WEAKLY ACIDIC (WITHOUT SLUDGE DISPOSAL)

	IRON (mg/l)	ACIDITY (mg/l)
HIGHLY ACIDIC	900-1200	2800-4800
MODERATELY ACIDIC	600-700	1400
WEAKLY ACIDIC	322	600-700

BASED ON DOYLE, BHATT, AND RAPP (1974)

Figure 36. Total operating cost (including capital costs) - hydrated lime treatment.

operating costs, respectively, against plant capacity for different acidity levels. On these figures, data for the study mine have been added to the reference data.

On Figure 35, the solid-line plots for 200 mg/ℓ through 5000 mg/ℓ acidity are original Operation Yellowboy data. At the study mine the exact quality of the input water to the treatment plant is not known. It is a mixture of approximately 6700 m^3/d (1.78 mgd) of 105 mg/ℓ acidity and 50 mg/ℓ total iron with 5000 m^3/d (1.32 mgd) of 15 mg/ℓ alkalinity and 12 mg/ℓ total iron, but the proportions of the mix vary considerably and nonpredictably. It may be conservatively assumed that the mixed input is 11 700 m^3/d (3.1 mgd) of 100 mg/ℓ acidity and 34 mg/ℓ total iron. A dashed line representing study facility flow of 100 mg/ℓ acidity has been added to Figure 35 using a ratio method of extrapolation. This level of acidity is considerably lower than the low end of the range of reference data (200 mg/ℓ).

The study mine's design capacity versus capital cost has also been plotted (circle) on Figure 35. The construction cost for the study mine treatment plant (maximum capacity of 19 000 m^3/d--5 mgd) is $430,000 (Table 17), or $423,000 in 1972 dollars. Since the acidity level of the water input to the plant was very conservatively estimated at about 100 mg/ℓ (dashed line) and the expectable acidity according to the reference data is about 89 mg/ℓ, then the design cost of the plant appears to be within the range of typical values, and perhaps on the low side.

Figure 35 serves to compare the operating costs of the study mine treatment plant with others in the industry, as represented by Operation Yellowboy models. Treatment costs (including sludge disposal) at the study mine are 4.9 ¢/m^3 (18.5 ¢ per 1,000 gal--see Table 19), or 4.8 ¢/m^3 (18 ¢/1,000 gal) in 1972 dollars. This value has been located on Figure 34 at the average daily treatment flow of 11 700 m^3/d (3.1 mgd). Again, this value is compatible with the reference data.

Table 22 presents cost contributors to annual operating expenses as percentages of total for the study mine facility; similar percentage costs have been extrapolated from the Operation Yellowboy data to model a roughly comparable facility. Differences in accounting systems and differences in operations limit the amount of detail that can be achieved in this comparison. Initially it seems noteworthy that only the costs for lime appear to be equivlent--a little less than 20 percent of the total. It is not possible to make a direct comparison on labor and repair costs, since these items are separated in the Operation Yellowboy accounting procedures, but in the study mine accounts they are both aggregated along with materials. Depreciation of the study plant facility is calculated at 5 percent, based on a 20-year cycle. The percentage plant cost for the Operation Yellowboy facility seems anomalously high (43.6 percent), but apparently reflects the fact that all the studies were conducted in a single, rather elaborate facility with a configuration that could treat both weakly and highly acidic water at large and small flow rates. This facility presumably has greater automatic control than does the study facility. However, the study facility employs more labor, undertakes more repair, and is undergoing minor improvements to upgrade its operation. These costs at the study mine facility considerably inflate the

total percentage for labor, materials, and repair. It may be noted that if the percent costs for these items at the study mine facility are added to the percentage for depreciation, the total (64.5 percent) is approximately comparable to the total of percentages for Operation Yellowboy labor, repair, and plant cost (63.2 percent). Sludge disposal cost for the study facility is about double that in the reference data because of truck rental expenses. Miscellaneous costs, on the other hand, are 75 percent higher for the more sophisticated Operation Yellowboy facility since these costs include water quality monitoring and other testing. Though such variations in the relative percentage components of annual operating costs are relatively large, in the long run they apparently offset one another.

It may be concluded that water treatment costs at the study mine facility (in terms of both capital outlay and yearly operating expenses) are approximately representative of industry costs for a plant of its capacity and for water of this acidity level.

COSTS OF MINE DEWATERING USING PUMPED WELLS

The costs of mine dewatering using pumped wells are separated into operating expenses and capital investment. These are further segregated and costed for electrical service (at the same rates as charged at the study mine), maintenance labor and overhead, operation labor and overhead, and depreciation of capital investment. All capital costs are depreciated over ten years (straight line). Included as capital investment are maintenance parts, well construction, pumping facilities, and engineering.

Three dewatering wells 152 m (500 ft) deep are costed, which represents the number and the approximate depths of the wells in the pilot dewatering operation. These costs are subsequently used in the comparison with the mine cost reduction, represented by the amount of water intercepted before entering the mine. The major cost contributors are well construction (capital investment) and electrical service (operating expenses). The cost items contributing to capital costs are listed on Table 23 without identified subcosts. The operating costs are presented on Table 24.

To illustrate the effect of well depth on costs of dewatering, the capital and operating costs for one well were manipulated to correlate with well depths between 30 m (100 ft) and 305 m (1,000 ft). These data are graphically presented in Figure 37 (less $2,500 for access rights). The increased slope for capital investment after the 132 m (400 ft) depth is due to the change in well and casing diameters. The well and casing diameters are increased from 20 cm to 25 cm (8 in. to 10 in.) and 15 cm to 20 cm (6 in. to 8 in.) respectively.

Operating costs increase linearly with increased depth. To dewater different mines, different numbers of wells of various depths would be required. Figure 38 graphically presents the variation in annual operating costs with number of wells for well depths of 30.5 m (100 ft), 61 m (200 ft), 122 m (400 ft), 152 m (500 ft), 183 m (600 ft), 244 m (800 ft), and 305 m (1,000 ft). Figure 39 shows capital costs versus number of wells and well depths. The

TABLE 23. CAPITAL COSTS FOR THREE WELLS 152 m (500 ft) DEEP

Cost Item	Cost
Well Construction (total cost)	$34,269
Fringe, overhead, administration	
Drilling	
Well Development	
Casing (material and labor)	
Expenses at site	
Mobilization	
Access Rights	2,500
Pumping Facilities (total cost)	10,794
Pump	
Discharge pipe, cable	
Pump installation (labor)	
Wire pump to electrical source (labor)	
Engineering (15% of above less access rights)	6,759
Labor	
Fringe, overhead, administration	
Total	$54,322

costs increase directly in proportion to the number of wells. The major cost contributors (well construction in capital costs, and electrical service in operating costs) maintain their percentage of costs with an increase in number of wells.

COST-EFFECTIVENESS COMPARISONS

Costs have been developed in the foregoing discussions for the present system of mine drainage and treatment, and for dewatering from pumped wells. In this segment, cost comparisons are presented for the effectiveness of the pilot dewatering operation and for projected full scale dewatering of the presently active part of the mine.

TABLE 24. ANNUAL OPERATING COSTS FOR THREE WELLS 152 m (500 ft) DEEP

Cost Item	Cost
Electrical Service	$30,606
Maintenance	3,870
Labor Fringe, overhead, administration	
Operation	5,832
Labor Fringe, overhead, administration	
Depreciation of Capital Costs	5,184
(10-year life, straight line, less access rights)	
Total	$45,492

Cost-Effectiveness Based on Pilot Dewatering (3 wells)

A comparison is made between the reductions in both annual operating expenses and capital investment costs vs. the cost of pilot dewatering (3 wells). In the existing water handling system, mine drainage and water treatment both contribute to annual operating expenses and to capital investment; well dewatering includes operational and investment costs of wells only, because no treatment is required for the water discharged from the wells. The method used is to estimate the reduction of mine drainage and treatment costs from the ratio of the inflow reduction to the average annual mine drainage.

The average mine inflow reduction in the Main G study area was 2.33 ℓ/s (37 gpm), or 200 m^3/d (0.053 mgd) during the pilot dewatering period, as opposed to an average annual flow of 11 700 m^3/d (3.1 mgd) for mine drainage. The reduction is 1.7 percent of the total mine drainage. This percentage is projected to an annual reduction and applied to the values on Table 17 (last column) for annual operating expenses and capital investment; the resulting reductions are then compared with dewatering costs (from Tables 23 and 24).

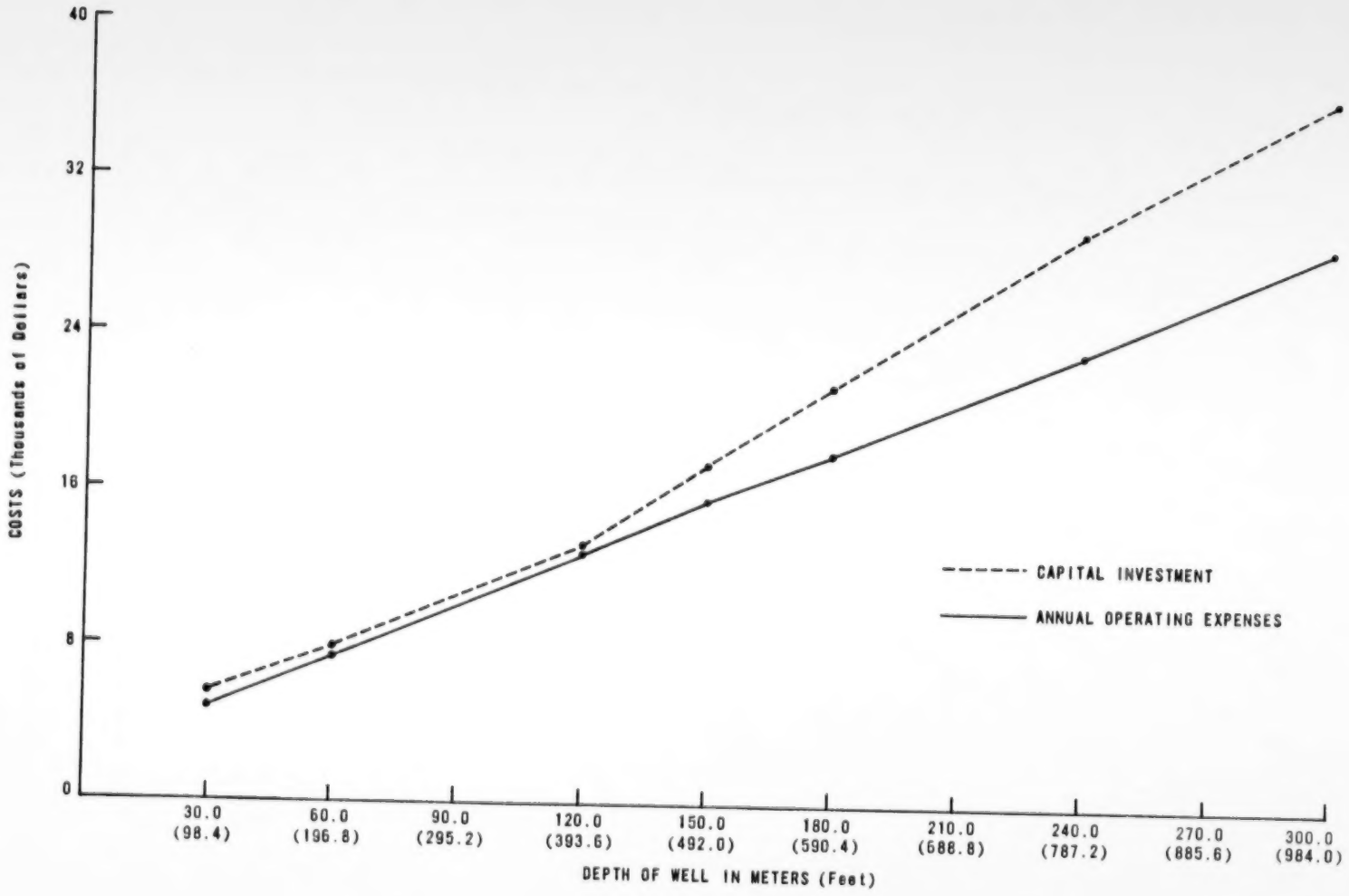


Figure 37. Operating and capital costs per well vs. well depth.

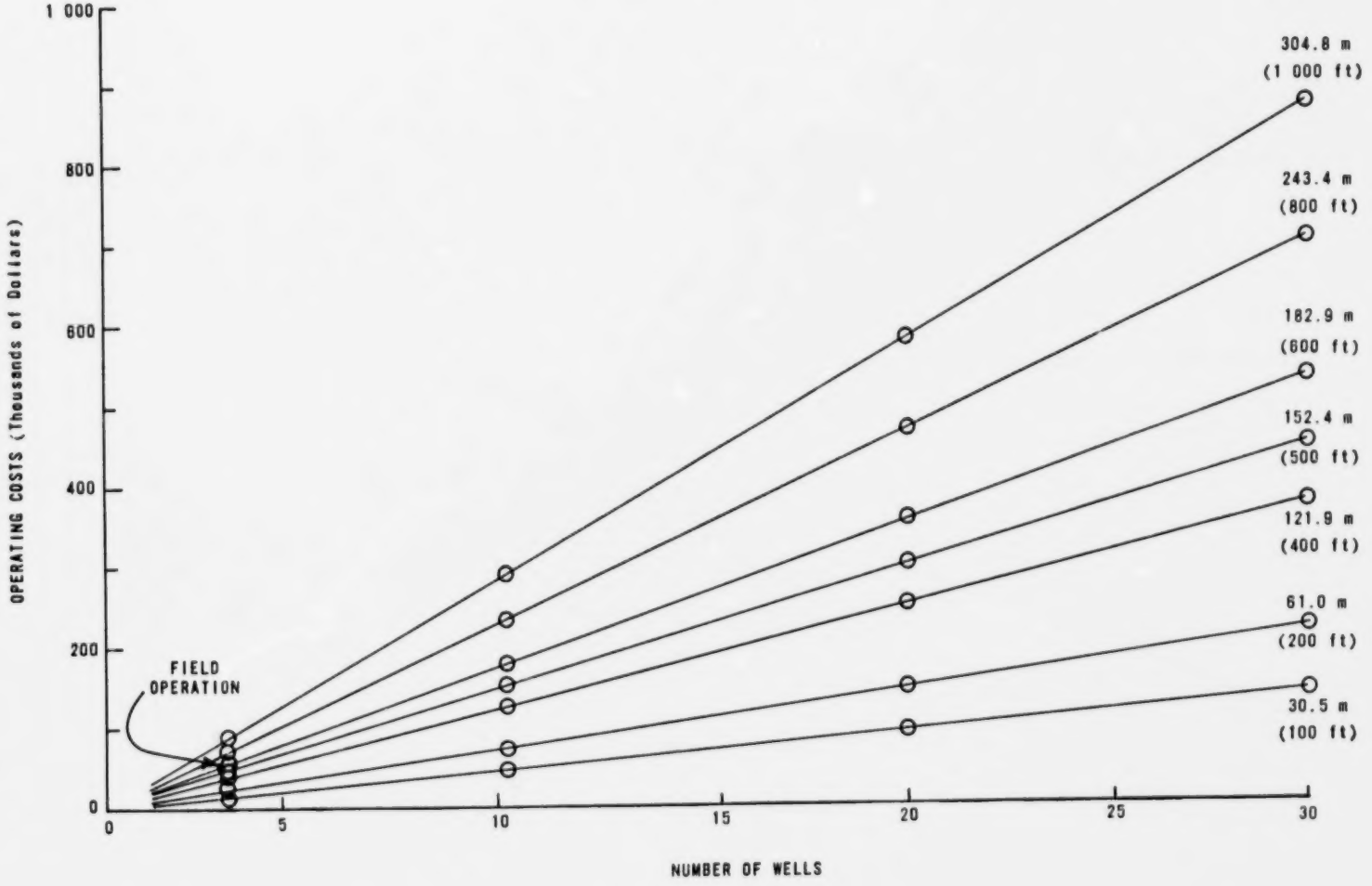


Figure 38. Annual operating costs vs. number of wells and well depth.

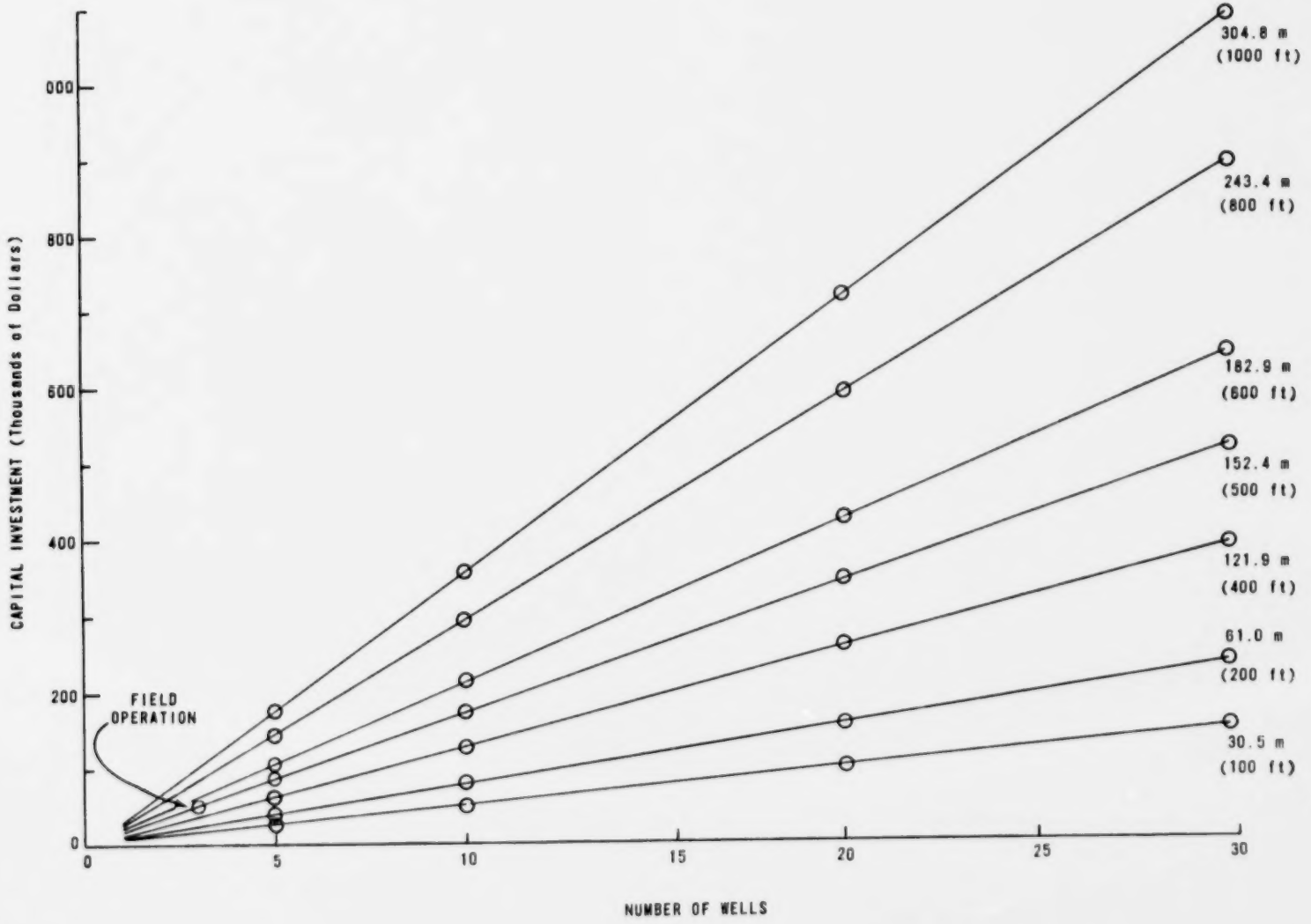


Figure 39. Capital investment vs. number of wells and well depth.

These costs and the comparisons are shown on Table 25. The reductions of drainage and treatment costs based on the 45 percent average well-effectiveness ratio of reduction of mine inflow to total well yield obtained during the pilot dewatering period, are $12,600 in annual operating costs and $13,100 in capital costs. These cost reductions are then divided into operating and capital costs for the three dewatering wells, resulting in cost ratios of 3.6 and 4.1 respectively. This indicates that the three-well dewatering system, at 45 percent effectiveness, is 3.6 times greater in annual operating costs and 4.1 times greater in capital costs than the corresponding savings it produces.

As explained previously in Section 7, the well-effectiveness is estimated to range between 50 and 80 percent depending on pumping time and recharge events. Theoretically, it should approach 80 percent effectiveness after 120 days of pumping providing there are no major recharge events and no recharge boundaries are intercepted. These levels of 50 and 80 percent effectiveness would result in a decrease in mine drainage by 1.9 and 3.1 percent respectively or 220 m^3/d (0.059 mgd) and 360 m^3/d (0.095 mgd). As shown on Table 25, at 80 percent well-effectiveness, the three dewatering wells are 2.0 times more costly in terms of annual operating costs and 2.3 more costly than capital costs of the existing system.

TABLE 25. PILOT DEWATERING COSTS
VS. MINE DRAINAGE AND TREATMENT COSTS

Item	Annual Operating Costs	Capital Investment
Total Drainage and Treatment Costs ($1,000)	739	772
Dewatering Costs-3 pilot dewatering wells ($1,000)	45	54
Reduction of Drainage and Treatment Costs ($1,000)		
45% well effectiveness (pilot dewatering period)	12.6	13.1
50% well effectiveness	14.0	14.7
80% well effectiveness	22.9	23.9
Ratios of Dewatering Costs to Cost Reductions (3 wells)		
45% well effectiveness	4.7	4.1
50% well effectiveness	3.2	3.7
80% well effectiveness	2.0	2.3

In the same manner, costs can be projected for dewatering the entire active area of the mine, as shown on Table 26. The drainage from the active area is estimated to be 60 percent of the total mine drainage or 7000 m^3/d (1.86 mgd). Assuming an average well yield of 1.9 ℓ/s (30 gpm) at 80 percent effectiveness, it would take 54 wells to completely dewater the active area. From Figure 37, the annual operating cost would be $820,000 and from Figure 38, the capital cost would be $930,000. Dewatering with these 54 wells would reduce drainage and treatment costs by $443,000 in annual operating costs and $463,000 in capital costs. The cost of dewatering with these wells would be 1.9 times more expensive and 2.0 times more expensive than the reductions in operating costs and capital costs respectively. Similarly, Table 26 shows that if the well-effectiveness is only 50 percent, 86 wells would be required and they would cost 2.9 and 3.2 times more than the reductions in annual operating costs and capital costs respectively. These ratios are slightly lower than the ratios for pilot dewatering (Table 25) because they reflect a slightly higher average well yield of 1.9 ℓ/s (30 gpm) than experienced in the pilot operation. This higher well yield should be easily attainable with better well design, well siting, and pump selection. The costs of dewatering reflect these improvements.

These calculations all indicate that dewatering with vertical pumping wells would be at least twice as costly as current mine drainage and water treatment at this mine and that dewatering apparently would not be a cost-effective means of controlling AMD in this particular situation. However, this does not mean that a dewatering system of this type is not economically

TABLE 26. ACTIVE MINE AREA DEWATERING COSTS VS. MINE DRAINAGE AND TREATMENT COSTS

Item	Annual Operating Costs	Capital Investment
Total Drainage and Treatment Costs ($1,000)	739	772
Dewatering Costs ($1,000)		
80% well effectiveness w/54 wells	820	930
50% well effectiveness w/86 wells	1,290	1,470
Reduction of Drainage and Treatment Costs for 60% of total mine drainage handled by dewatering ($1,000)	443	463
Ratios of Dewatering Costs to Cost Reductions		
80% well effectiveness	1.9	2.0
50% well effectiveness	2.9	3.2

feasible. If fracture zones can be effectively located by exploration, average yield of dewatering wells can be greatly increased. An increase of three to four times the average yield (1.9 ℓ/s--30 gpm) used in this study or 5.7 to 7.6 ℓ/s (90 to 120 gpm) per well would make this type of system cost-effective at the study mine, both in terms of operating costs and capital costs. There are other benefits to be derived from mine dewatering that should be considered. Loss of mine production from unstable roofs and sudden large inflows of water can be reduced or possibly eliminated by dewatering. In some mines, increased production could therefore be the largest single benefit derived from mine dewatering.

Other potential benefits may include more opportunities to use the water pumped from the dewatering system by overlying and adjacent communities or land owners. Also, the water discharged from wells is better in overall quality (at least in the case of this study) than treated water. It has less total dissolved solids and sulfates than the effluent from the treatment plant. Although it would be difficult to place a dollar value on this benefit, it could have only positive effects on agricultural and consumer water applications. There are probably many other benefits to the mining operation. All of these potential benefits will vary widely from mine to mine, and the economic feasibility of mine dewatering must be considered separately for each case.

Effect of Water Quality on Dewatering Costs

From the foregoing discussions, it is apparent that dewatering would be cost effective if the quality of mine drainage was so poor that the cost of treatment would offset dewatering costs. By using the ratios developed for full scale dewatering shown on Table 26 for capital costs, the capital treatment costs and annual treatment operating expenses required to break even with full scale dewatering can be calculated. The mine drainage costs are assumed to remain the same and it is assumed the well discharges would not require treatment. The break-even capital investment for treatment using a well-effectiveness of 80 percent would equal (2.0 x $772,000) - $342,000 or $1,202,000. Plotting this value on Figure 33 for a capacity of 11 700 m^3/d (3.1 mgd) gives an acidity of about 500 mg/ℓ. The corresponding break-even operating treatment expense would be (1.9 x $739,000) - $530,000 or $874,000. This value equals 7.4 ¢/m^3 (28 ¢/1,000 gal), and when plotted on Figure 34 falls slightly under the weakly acid curves (600-700 mg/ℓ).

For a well-effectiveness of 50 percent, the break-even capital investment cost is (3.2 x $772,000) - $530,000 or about $2,128,000. Plotted on Figure 35 for a capacity of 11 700 m^3/d (3.1 mgd), this value results in an acidity of about 1100 mg/ℓ. The break-even operating cost at 50 percent well-effectiveness is (2.9 x $739,000) - $430,000, or about $1,613,000, which reduces to 13.7 ¢/m^3 (52 ¢/1,000 gal). This value is closest to the moderately acidic curves on Figure 36. These calculations indicate that dewatering would be cost-effective if the acidity ranges between 500 and 700 mg/ℓ and the dewatering wells are 80 percent effective in intercepting mine inflows. If the

dewatering wells are 50 percent effective, the acidity would have to be in the 1000 to 1500 mg/ℓ range. Again, if other benefits such as roof stabilization are of importance, it may be economically feasible to dewater with substantially lower acidities than the ranges estimated here.

Effect of Mine Depth on Dewatering Costs

The depth of the mine would affect the costs of dewatering primarily due to well depth and pump size requirements (capital costs) and energy requirements (annual operating expenses). Mine costs would also change because of different pump sizes required at the mine exit stations (capital cost) and because of different energy requirements (operating expenses). Projections of data developed herein for the Lancashire No. 20 mine revealed that dewatering would not be cost-effective unless the mine was less than an average of 67 m (220 ft) deep when the average dewatering well-effectiveness was 80 percent. For a dewatering well-effectiveness of 50 percent, the mine has to average less than 30 m (100 ft) deep. The divergence between dewatering costs and reductions of drainage and treatment costs is rapid with depth, and other benefits must be found to make up the differences. It does suggest that the shallower the mine, the more serious the consideration that should be given to mine dewatering for water quality control.

Effect of Dewatering System Design on Costs

There are other dewatering systems that can be designed which appear to be less costly than the system used during this study. Vertical wells can be drilled from the surface into mine openings and the water in each well can be allowed to drain by gravity into a header pipe. The water is then transferred to a central pumping plant from which it is pumped to the surface for discharge. The capital costs of this system would be about 15 to 20 percent less because only one pump is used for a group of wells (or possibly all wells) and surface access would be less costly. The operational costs would be about the same as for individually pumped dewatering wells, assuming the same well-effectiveness and water qualities as before. It is readily apparent that this type of system would not be cost-effective for control of AMD at the Lancashire No. 20 mine.

A dewatering system that appears to be cost-effective would be a system of inclined drainage holes 6.4 cm (2.5 in.) in diameter, drilled with conventional mine equipment. The drilling machine would be mounted on a mobile column platform and would use 0.19 m^3/s (400 cfm) of 690 x 10^3 Pa (100 psi) air to drill with. Capital costs were estimated on a footage basis and are summarized in Table 27. The drain holes would be drilled into the mine roof from entries around the periphery of the mine and along longwall panels at angles that would intersect fractures and drain by gravity into a header pipe.

TABLE 27. CAPITAL COSTS FOR INCLINED DRAINAGE HOLES

Item	Cost (per meter)	Cost (per foot)
Equipment	$0.85	$0.26
Labor	2.16	0.66
Expendable Materials	2.95	0.90
Air, oil, etc.	1.02	0.31
Bits	0.26	0.08
	$7.24	$2.21

From the data available, there is no way of predicting the flow from these holes, but a break-even yield can be calculated for capital costs in terms of flow per 30 m (100 ft) of drain hole. It can be assumed that the collection of water from these holes into header pipes and eventually into a sump or sumps for pumpage to the surface would cost about the same as current underground water collection. Therefore, the cost reduction would be in treatment costs. From Table 17 the capital costs for water treatment are $430,000. At a cost of $7.24 per meter ($2.21 per foot), 59 300 m (194,600 ft) of drain hole can be drilled for about the same cost. The yield per 30 m (100 ft) of drain hole to handle 11 700 m^3/d (3.1 mgd) or 135 ℓ/s (2,153 gpm) would be 135 ÷ 1946 or 0.07 ℓ/s. This yield per 30 m (100 feet) of drain hole appears to be a reasonable yield to attain and it could easily be exceeded if the drain holes are positioned to intercept fracture zones or major joint sets. If the cost of well installation and operation only breaks even with capital costs for water treatment, there would still be a savings in operating costs for water treatment. This system is flexible and discharges can be made to the surface through boreholes where convenient. Actually this system could result in additional savings in water collection costs since it would require less piping, and other mine drainage that is good quality could be combined with the drain hole water and discharged without treatment. Also, this system is not sensitive to mine depth because the cost of drainage holes would be the same for all depths. Therefore, this system compares favorably with the existing system and appears to be a cost-effective approach, especially if other benefits can be realized.

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